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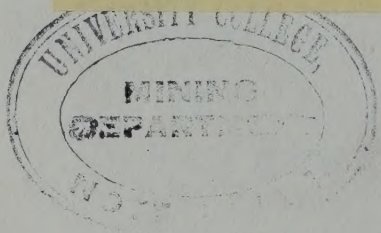

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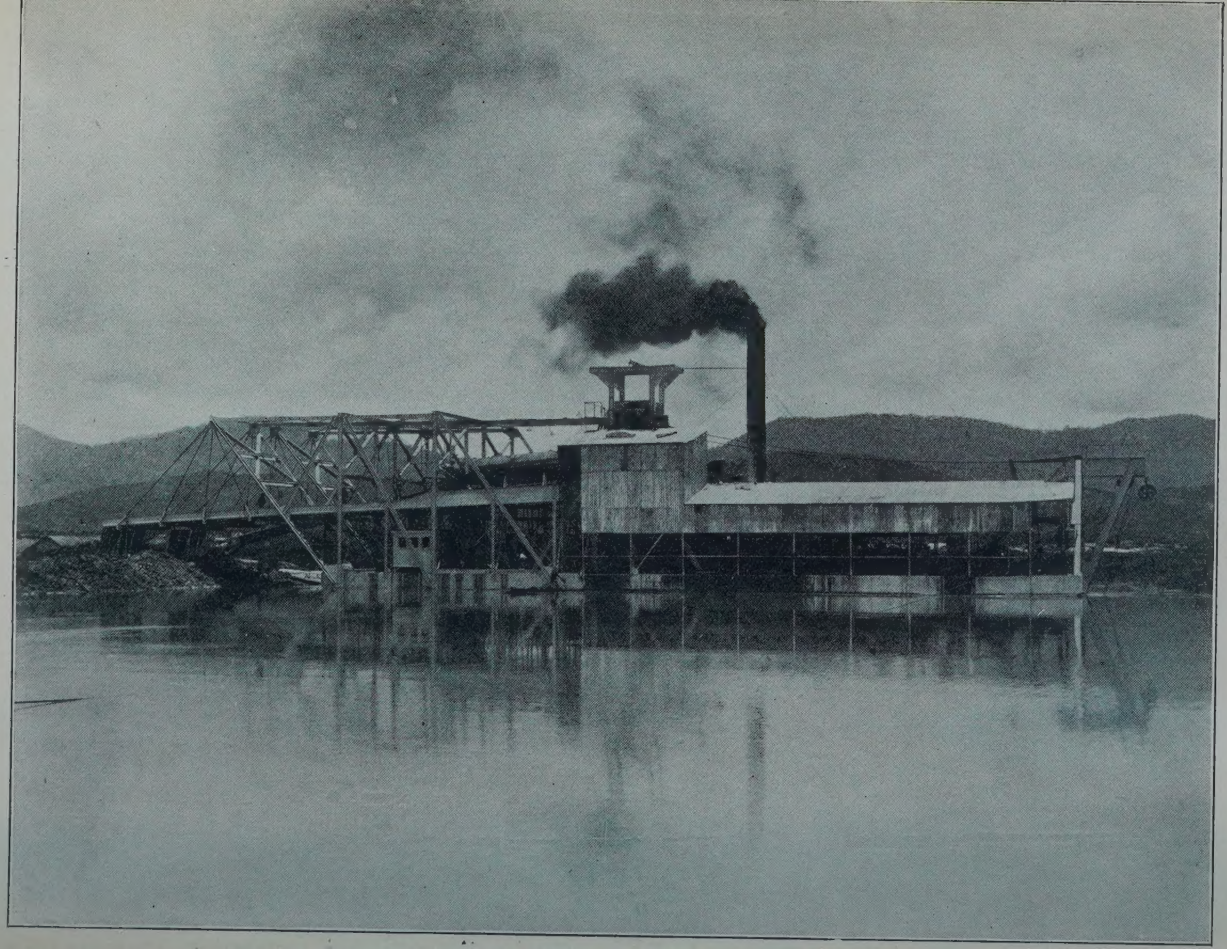

MINING OF ALLUVIAL DEPOSITS BY DREDGING AND HYDRAULICKING

BY

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PREFACE.

In preparing these notes we have endeavoured to present the practical side of the work met with in the exploration, prospecting, and management of alluvial propositions by dredging and hydraulicking, and they represent for the most part our experience in all parts of the world.

We have touched on the theoretical side in only those instances where we considered that a more detailed explanation would meet the requirements of students without field experience and those practical men who have not had the advantage of a technical training. This refers especially to problems connected with the measurement and use of water. We have refrained as much as possible from the use of such vague terms as the " Miner's Inch " and adhered to the expression of quantities of water in cubic feet per second. Examples of working instances have been given to make the text clearer for the practical worker and as many drawings and photographs furnished as was considered commensurate with a work of this size.

We venture to hope that much of the information contained herein will be of practical use to our fellow engineers and will gladly welcome any suggestions from them for the improvement of the work or the correction of any errors in the text.

Our thanks are due to Miss N. Warner Hooke for her execution of most of the drawings and we acknowledge the very great help we have received from the publishers.

We also tender our thanks to the following firms for the various blocks, photographs, and information so generously supplied by them :—The Climax Rock Drill and Engineering Works, Ltd., The Lea Recorder Co., Ltd., Messrs. Ruston and Hornsby, Ltd., and Messrs. Werf Conrad.

London,
December, 1929.

W. E. THORNE.
A. W. HOOKE.

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PART I.

INTRODUCTION.

The recovery of minerals from alluvial deposits ranks among the oldest methods of mining and the art has been practised by the human race in most parts of the world. Gold has been more sought after than other minerals owing to its wider distribution, its demand for ornamentation, and the relative ease of its recovery from placer deposits. This is due to its high specific gravity, its occurrence in the free state and in rich deposits, and to the low cost of equipment. It becomes thus the favourite means for quick wealth for the poor man.

After the various recovery methods such as panning, rocking, ground sluicing and hydraulicking whereby the more readily accessible deposits were worked, came the practice of dredging. This has greatly improved from the first very crude structures to the better machines of to-day and improvements are still being made, for seldom are two dredges constructed exactly alike, except in the design of the main standard parts—which should always remain the same for various reasons which will be gone into more fully later.

Among alluvial deposits tin follows gold in economic importance, then platinum and diamonds. Other minerals have been won from placers but are not of such importance and their method of recovery follows with little variation the processes adopted in winning gold or tin.

Alluvial Mining.—The term " Alluvial " or " Placer " mining is usually applied to any method of working which has for its object the extraction of such minerals as those bearing gold, tin, or platinum, and such gems as diamonds, from the drifts in valleys or terraces which have been formed by the denuding forces of nature and may be said to have begun with the recovery of alluvial gold. The process of gold extraction has varied and improved as time went on, and in later years been applied in modified forms to win such other minerals as those enumerated. In fact the term alluvial mining is now accepted in the broadest sense to embrace all the methods adopted in winning such minerals, ores or gems as occur in nature in a disintegrated state and which are merely loosely associated with gravel, earth or clay.

History.—The authors know of no definite record of where the first placer mining operations were conducted. They have examined old workings in the Andes that are supposed to have been worked for gold some five thousand years ago and others in Rhodesia from which King Solomon received a part of his gold. In Spain mines that were worked before the Christian era by the Romans and Brazilian diamond placers that have been producing for 180 years. The above workings were pits and ground sluicing. The first hydraulicking was done about seventy-five years ago. The first successful dredges were of the spoon type, built in 1865. These were of varied design, scoop or shovel, manipulated from rafts and pontoons, and were profitable. They were soon superseded by the bucket elevator type designed by Scott of New Zealand whose machine was the forerunner of the present day modern bucket dredge.

Methods of Gold Recovery.—The Incas of South America sank pits or shallow shafts, generally along the outcrop of the reef or vein, in the detrital matter, or in the shallow gravel. No indications were observed that any excavation exceeded twenty feet in depth. The method of recovery is not known but it is presumed to have been by a wooden dish or " calabash ", possibly some use being made of a cocoanut shell. Quite a large amount of gold, according to their standard of wealth, must have been won, as history shows that the Spanish received considerable amounts from that source over a number of years, and to-day gold ornaments are recovered in the process of ground sluicing, hydraulicking and dredging in that country. Some of these ornaments are found on the bedrock of gravel deposits 20 to 30 ft. deep, as it was the practice of these people to bury the ornaments with their dead along the valleys. The movement of the stream from side to side across the valley has brought the gold ornaments into the recent river gravels from which they have since been recovered. It is no mere theory that the ornaments were placed in the graves, for the authors have assisted in their recovery from such places and possess quite a few at the present time.

In Africa they have observed the old workings in the detritus of outcrops, and in the alluvial. Recovery here was probably made by the use of the calabash.

The Roman workings in Spain were generally operated by ground sluicing. Ditches were cut from a stream and conducted along the hillside to the top of the bank to be worked and then down to the " face ", and work was carried on much the same as it is to-day. The method has not been improved upon to any extent. As the gravel was worked the boulders were stacked on the cleaned bedrock. Most of the material treated here was glacial debris, some of which has been re-arranged by the present-day streams.

The authors believe the Batea was used for handling the concentrate from the ground sluice. As the work was done by slaves, the operating costs must have been low. Old watch towers are still to be seen, from which guards kept a look-out on the workers. On sampling the " faces " where the workmen left off, the average value was not above one penny per cubic yard. As quite a bit of clay was in the deposit it probably did not pay to operate and, also, lack of sufficient gradient and dumping ground appear to have been against continuance.

Hydraulicking.—The method of hydraulicking originated in California and was brought about by such favourable conditions as deep deposits, suitable dumping room and gradient, good water supply, a sufficient gold content and an all-year working season. It has the lowest operating cost of any method evolved up to the present time, such having at times been as low as three farthings ($1\frac{1}{2}$ cents) per cubic yard. This figure, of course, does not cover amortization, but under proper conditions this is low, particularly when only short ditches and pipelines will assure an ample supply of water under an effective " head " or pressure.

The method is used for gold, tin and platinum. The authors do not know of any instances where it has been applied for diamond mining and do not think this application practical, owing to the low specific gravity of the diamond, and the difficulty of handling the large quantities of material usually dealt with in hydraulicking.

TYPES OF DEPOSITS.

Alluvial drifts are formed from glacier deposits and partially from the *débris* of large landslips in mountainous country which has been carried down by the streams and deposited in the valleys or flats along the course of the rivers to the sea, or by the general processes of weathering and denudation. The streams have been the medium for transporting and distributing the weathered and denuded material. The action that has formed the terraces and flats is still going on, and continues to form beds of gravel and sand in the course of the streams. The lighter material is carried forward in varying degree according to the velocity of flow, while the heavier material and metallic substances are gradually left behind. Thus we find values running for a considerable distance and confined to a certain width. It is alone due to the action of moving water that leads of gold or other minerals in drift terraces have been formed. In fact the streams are nature's ground sluices in which the material has been prepared and preserved for the more intensive methods of man.

A deposit may often be the re-concentration of an older and larger deposit, and the type to which it may belong is governed

largely by its original formation and the subsequent earth movements. A large slow-moving river meandering undisturbed for thousands of years across a wide valley would form a deposit quite distinct from that of a narrow fast running stream debouching from the mountains on to the plains below, or running into the sea.

Shallow or Modern Placers.—The types embraced under this head are alluvial deposits that are most accessible, such as those occurring in the adjacent old courses and the present beds of recent streams, at river bars and on the sea-shore. Hillside or bench deposits which are really the remains of older deposits might be classed under this head. It will be evident that in this, as in most other classifications, intermediate types are certain to be found which may belong to either one group or another. Hillside placers, for example, are those that occur on the slopes of hills and do not occupy a defined channel. These, though water sorted to a certain extent, merge directly into deposits of residual origin and into stream deposits.

Deep Leads or Ancient Placers.—This is a term applied to the deeper and more particularly the " capped " deposits, i.e. deposits which, since their formation, have been covered by a volcanic flow. Deposits in some cases have attained great depth due to excessive deposition of material. The rivers which formed them have been diverted from their original courses either by silting up, changes due to earth movements, or by volcanic eruptions blocking their course and covering the deposited material with a sheet of lava which may vary in thickness from a few feet to hundreds of feet. The gold deposits of the Dargo High Plains in Victoria and the tin deposits of the Ringarooma Valley, Tasmania, are examples of this type of deposit.

PROSPECTING AND VALUATION.

Except in the case of shallow and more readily accessible deposits, the prospecting and testing of an area is often beyond the financial resources of the discoverer or an individual and at this stage it is generally considered sufficient to sink a few pits or boreholes at widely spaced intervals to obtain an idea of value and depth as a preliminary to seeking the assistance of some finance corporation. This being accomplished, the next step is a thorough examination of the property in detail. This work should always be under the direction of a qualified engineer of repute.

With a preliminary examination of the ground and a regard to physical and geological conditions, his experience will enable him readily to conceive a plan for the lay-out of the work that will accomplish most at a minimum cost. Such an examination and

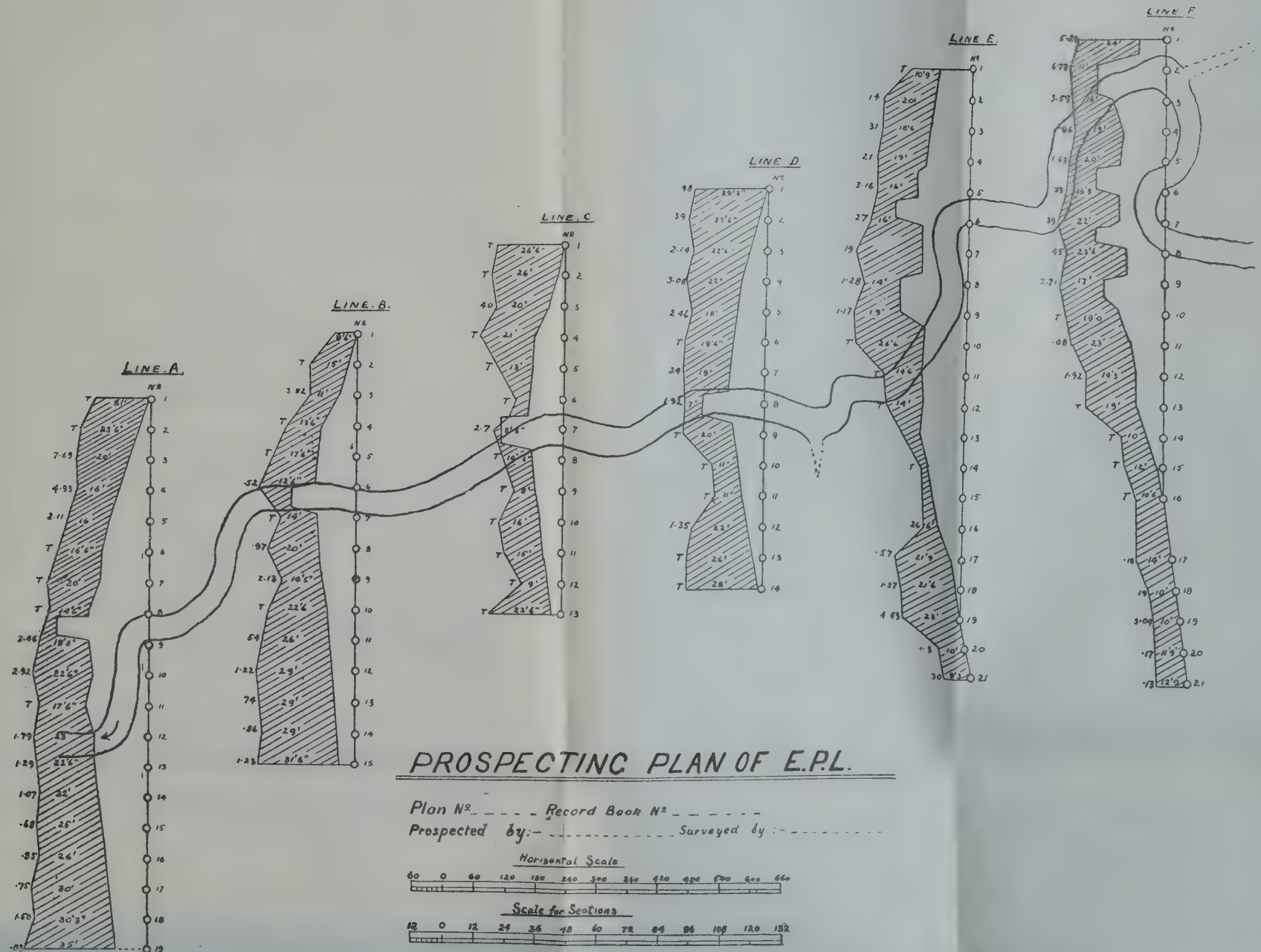
absorption of the data available is time well spent in any circumstances. No two deposits are identical and a careful reconnaissance is the first step in diagnosis and often a potent factor in successful development. Never go on to a property with a pre-conceived policy. Approach the work with an open mind and use your training and experience to guide you in a complete interpretation of the results as furnished by the pitting and boring. A close study and analysis of all conditions is worth while, and a general tour around a property before any work is undertaken is time very well spent.

Prospecting.—The importance of careful prospecting is particularly emphasized because probably 80% of the failures in alluvial mining are primarily due to inadequate or inefficient testing work, or to neglect fully to appreciate the influence of all the physical features. A very high grade deposit may justify some neglect of the latter, but in deposits where the gold or tin content per cubic yard is not high, it may only be possible to work the property profitably by giving the fullest consideration to all topographic features and the exercise of the most skilful mining methods. Topography covers a multitude of controlling factors :—" Location " will influence the cost of transport which is an important item. " Climate " will dictate the length of the operating season in arctic and sub-arctic regions, and the absence of rain may determine the extent of operations in tropical countries with pronounced seasonal conditions. The absence or abundance of timber will often seriously affect conditions. Topography will affect the gradient of the streams, the facilities for the easy disposal of tailings, the construction of dams for water storage, and the lay-out of ditches for the conveyance of water. A full conception of all these points has the closest relation to mining costs. In lode mining it is almost impossible to determine the value of the deposit without a large amount of costly development work. In alluvial mining, however, modern methods have made it possible to determine with reasonable cost and close accuracy the value of the mineral in the deposit. The two accepted methods of making this determination are by drilling or pitting. In less frequent instances deep shaft-sinking or tunnelling may accomplish the desired result. The whole of the sample obtained in the drill core is treated, but it is rarely the practice to treat the whole of the excavated material from a pit, although this procedure has distinct advantages in some cases and is recommended in very stony ground or when there are a number of large boulders. A sample-cut of regular width and depth is usually made down the side of the pit from top to bottom, the material therefrom forming the sample for estimation. Such a sample may not exceed in volume that of a core from a borehole of the same depth, but

check sampling is possible in the case of a shaft, whereas with the drilling method a new borehole is necessary. Even when a property is tested by drilling it is advisable, when possible, to sink an occasional shaft because it affords the engineer the opportunity of examining the deposit *in situ*. He can view the gravels in section and note the proportion of coarse gravel and boulders (useful knowledge in the case of gravel pumping); also observe the nature and condition of the bedrock. Such pits if sunk around a borehole afford an opportunity of checking the drilling calculations. Bedrock is usually decomposed and it is occasionally good practice to sink into it several feet to ascertain how far one can go before meeting hard rock. Such information will tell one what the probability is of being able, in the case of a sluicing proposition, to cut feed and tail races in the bedrock without recourse to blasting. This would also be useful information in the case of gravel pumping or dredging. Bedrock that can be readily cut into, allows of thorough cleaning up of the whole of the valuable mineral.

Prospecting by Drilling.—The hand-drilling equipment consists of a fairly stout steel casing, generally of 4 in. internal diameter made in 5 ft. lengths with a coupling that is flush inside and outside. The leading section is fitted with a cutting shoe which may have a plain or toothed cutting edge and slightly larger in diameter than the external diameter of the pipe or casing. The upper end of the pipe is surmounted by a platform on which the men stand and operate the drill. A variety of tools are used, the more common of which consist of the spiral auger, shell auger, chisel bit of varying design, various sand pumps, and sundry accessory tools, all of which in turn are attachable to the drilling rods which are usually 1¼ in. square section and 5 ft. in length. A short "matching" rod is sometimes used. This rod is of such a length as to make 5 ft. when one of the tools is attached. The operator is thus working in lengths multiples of 5 ft. and can readily ascertain the depth drilled.

At the beginning of operations a short entry-hole for the first section of the casing with its cutting shoe attached is generally made by an earth auger, unless the top layer of the ground is soft enough to permit the pipe to enter by giving it a turning motion. Its own weight, with that of the platform, will often do this. Once firmly entered, the operators can then mount the platform, which is turned by a lever (a horse gear is sometimes used for this) and there is then sufficient weight to cut into the ground. In very hard ground a driving cap is screwed on the top section of the pipe and a jarring tool or monkey is operated from the platform. This action, plus the weight of the rotating platform and men, will generally suffice to penetrate the toughest of ground. When the first section of pipe has entered 2 or 3 ft. the process of removing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LINE E.
LINE F
PROSPECTING PLAN OF E.P.L.
Plan Nº _____ Record Book Nº _____
Prospected by:- _____ Surveyed by :- _____
Horizontal Scale
60 0 60 120 180 240 300 360 420 480 540 600 660
Scale for Sections
12 0 12 24 36 48 60 72 84 96 108 120 132

FIG. 1.—SHOWING THE LAY-OUT OF DRILL HOLES OR PITS ALONG THE COURSE OF A STREAM. The bores or pits are shown in plan. The cross-sections show the contour of surface and bedrock and the figures respectively record the number of the pit or bore, the depth and value in lb. of tin oxide of 70% grade per cu. yd. In the case of gold or platinum the value should be stated in money (giving value per oz. on which calculation is based). In the case of diamonds, value is stated in carats per cu. yd.

[To face page 6.

the core begins. The method will vary according to the nature of the ground. If dry and firm this may be done by the spiral auger ; if of a looser nature the shell auger may be needed. If the ground is very wet a tool which simultaneously drills and pumps material into its barrel is often used. The casing can then be sunk and the material drilled and pumped in one operation. Care should be taken to keep the casing erect and the tools should never be allowed to cut the ground ahead of the casing. The casing should always be in the lead as it were. This assures of a plug of undrilled material being in the mouth of the casing and acting as a seal and preventing inflow from sources outside the pipe, particularly when in very soft or wet ground. If inflow occurs it will almost certainly vitiate results, and such work should be rejected and a new hole put down.

Fig. 1 represents a prospecting plan of drilling or pitting work along the course of a river. Lines of bores or pits are set out as shown in *A* to *F* at 400 ft. intervals with the boreholes 60 ft. apart and each line numbered consecutively. The hatched portions represent a cross-section of the ground penetrated by the various boreholes or pits and shows a profile of both surface and bedrock. Work recorded in this way is of particular value to the operator when subsequent mining operations begin. It not only shows the high sections of bedrock (useful information in the case of dredging) but will afford a valuable guide in the layout of the treatment plant by gravel pumping, ground sluicing or shovel excavators.

Reading to the left from the line of pits or bores, the figures quoted represent the number of the pit or borehole, the depth in feet and inches, and the value of the ground in lb. of black tin (70% metal) per cubic yard. In the case of gold this latter figure should be quoted in pence per cubic yard. All prospecting work should be systematically recorded and mapped and filed at the mine office. Neglect of this precaution has often led to a needless and expensive duplication of work. Valuable unrecorded work may be lost by the departure of an employee. Plans should be made and kept up to date. They have a greater economic importance in alluvial mining operations than is generally allocated to them by the average company director. The surveyor and his cost is too often and unwisely viewed as a luxury equipment.

Prospecting by Pitting.—In prospecting by pitting the haulage of the excavated material, and in very wet ground, the bailing of seepage water, are two factors which retard rapid work. Particularly is this to be noted where native labour is employed. The general method for ground of ordinary depth is by means of a bucket and rope but this entails a great physical effort on the

part of the natives. In the Malay States the astuteness of the Chinaman has overcome the burden of direct haulage by the adoption of a combination of bob-stick and whip-stick, the latter being suspended from the whip-stick by a short length of chain with a swivel link in the centre.

The illustrations in Fig. 2 show the side and end elevations of such a rig and give approximate dimensions for ground as deep as 16 ft. It will be readily seen, however, that greater depths

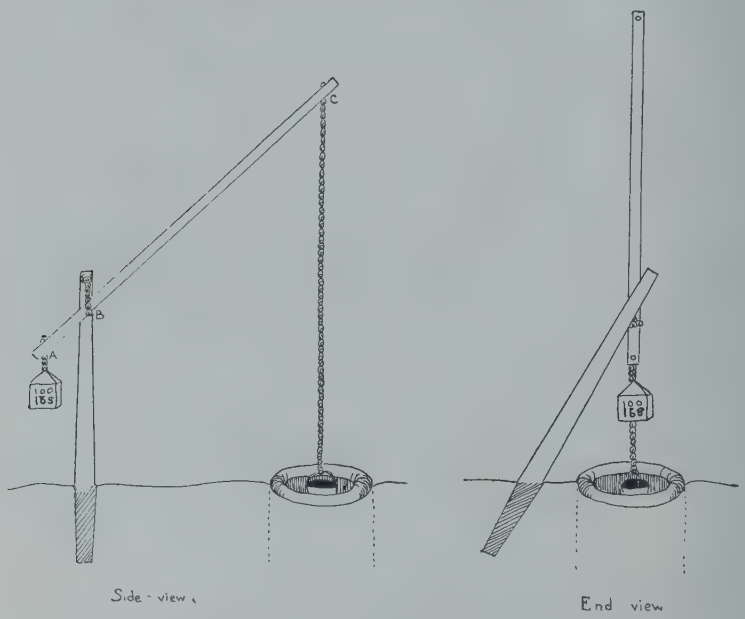

Fig. 2.—Illustrating the Bob-stick or Balance Pole used by Chinese as an aid in lifting Material from a Pit.

can be reached by increasing either the whip-stick or bob-stick or both, within reasonable porportions. The whole principle of the rig lies in the intelligent application of the lever and the fulcrum. The whip-stick being suspended as it is by a swivel chain allows of a horizontal movement over an arc of more than 180° and thus facilitates the dumping of waste or water well away from the collar of the shaft.

For deeper ground the ordinary miner's windlass is usually adopted. The authors have seen it used down to 250 ft. but consider that for depths over 100 ft. some type of power unit

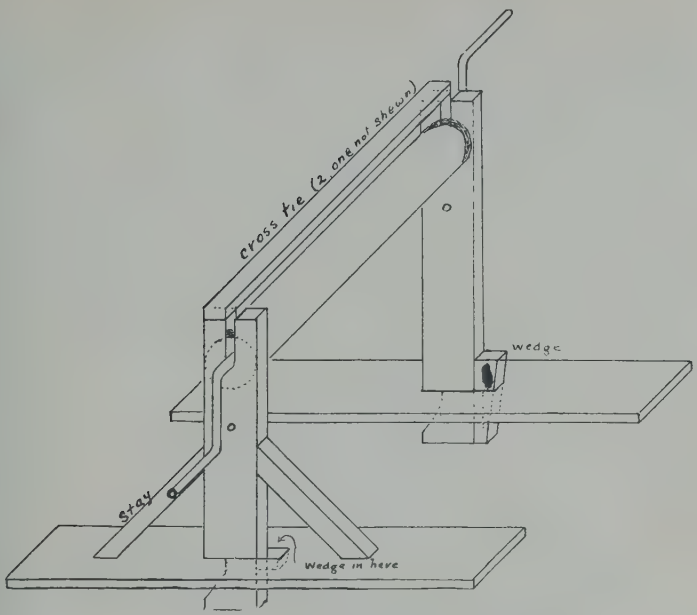

FIG. 3.—SKETCH OF SECTIONALIZED WINDLASS.

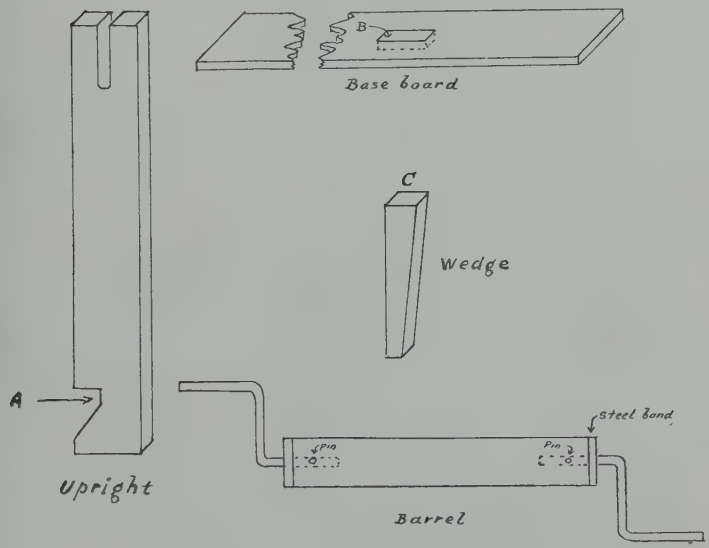

FIG. 3A.—PARTS COMPRISING THE WHOLE WINDLASS SHOWN IN FIG. 3.

or horse-gear will prove more efficient. A useful type of windlass is shown in Figs. 3, 3a, and 4. This can be readily dismantled and re-erected with ease by the ordinary unskilled native. The two uprights drop into the slotted holes in the base board and are then wedged into position by the wedges shown. The tapered section beneath prevents the upright from being lifted out by strains induced by the turning motion of the windlass barrel. The holes in the upright are for the insertion of a peg against which the handle of the windlass will press and be held in a locked position to prevent free rotation. Two battens along the top complete the outfit which is a light and rigid structure.

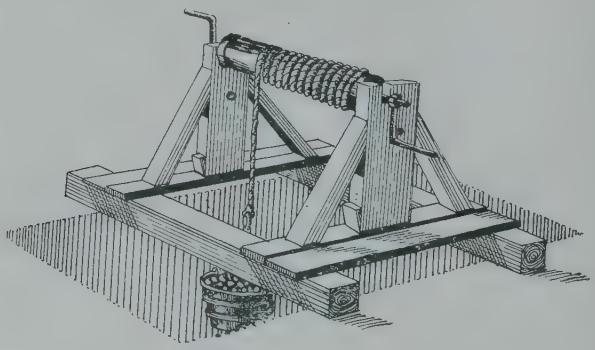

Fig. 4.—Windlass set over a Shaft.

Estimating Values.—Various formulæ have been designed for estimating the value of alluvial deposits from the results obtained either by drilling or pitting.

In the case of pitting the value of the ground is estimated in three ways in particular :—

(*a*) By treating the whole of the contents of the pit as a bulk sample and estimating on the cubic capacity of the pit.

(*b*) By taking a cut down the side of the pit, maintaining as evenly as possible a regular width and depth over the entire length of the pit, and then weighing the sample and calculating the weight on the basis of 3,000 lb. being equal to a cubic yard in place. This is the method accepted by the Institution of Mining and Metallurgy.

(*c*) By taking a cut of regular width and depth and calculating on the cubic capacity of the cut only.

Method (*a*) gives the most acceptable results as it more closely represents a working return, particularly where the ground contains a variable amount of stone.

Method (*b*) is not quite so close, as the inclusion of a large stone drawn out or left in, in the process of the cut, will partially vitiate the result.

Method (*c*) is not recommended, mainly because of the difficulty in maintaining a cut of even width and depth. Also in very wet ground in this case, and in the case of method (*b*), the incoming water may easily wash into the sample loose particles of gold, tin or platinum that do not rightly belong to it.

In the three instances enumerated the formulae would be as follows :—

(*a*) In calculating the value of the pit, the whole contents of which have been treated as a bulk sample, it only remains to divide the pounds clean tin recovered (milligrams in the case of gold or platinum) by the cubic capacity of the pit in yards to obtain the value per cubic yard. This is a simple method for the prospector.

(*b*) In the case of pitting where a sample cut has been made down the side of the pit, the whole of the sample is weighed and then carefully washed, and the results in the case of tin oxide calculated as follows: —

$$V = 6{\cdot}614 \frac{G}{Q}$$

Where Q = the weight of the sample in lb.
V = value per cubic yard in lb.
G = weight in grammes of clean tin recovered.

In this formula the weight of a cubic yard of material in place is taken as being equal to 3,000 lb. This is the accepted figure of the Institution of Mining and Metallurgy.

When gold-bearing gravels are being mined, the formula is as follows :—

Q = weight of dirt in sample in lb.
V = value per cubic yard in milligrams.
m = weight of gold in milligrams.

$$\text{Then } V = \frac{3{,}000 \times m}{Q} \times \frac{0{\cdot}030864}{1} = 92{\cdot}592 \frac{M}{Q}.$$

Note. One milligram of gold is worth 0·030864 pence with gold at £4 per ounce.

(*c*) Assuming that a sample cut of 2 in. by 2 in. or 4 in. by 1 in. has been taken, the formula for calculating the value per cubic yard is :—

$$V = \frac{0{\cdot}0348 \times A \times G}{L}$$

Where L = the depth of the pit in ft.
G = the weight of the tin concentrates in grams.
A = the percentage of metallic tin in the concentrates.

No formula calculated on the cubic dimension of a cut down the side of a pit can reasonably represent the value of the ground for the simple reason that a cut of even dimensions is almost impossible to make and introduces a personal element in the judgment of the depth and width of cut. Apart from this stones of varying size will be met which will be more than the dimensions of the cut and must be either included or excluded. The only way is to make the cut as even or regular as possible and weigh the sample and calculate on the basis of 3,000 lb. as representing the weight of a cubic yard in place.

Pitting has the advantage that it gives the operator the opportunity of observing both the nature of the ground penetrated and that of the bedrock. It also allows of check sampling.

In the case of drilling the results are calculated on the cubic displacement of the pipe casing estimated from the *external* diameter of the cutting shoe. The external diameter is taken because the cutting shoe is bevelled on the inner side and it is rightly assumed that any material cut by the edge of the shoe will be forced inwards by the bevelled face and become part of the sample. The acceptance of this method has been greatly contended, but the writers agree that it is the most representative and, in their experience, gives results most nearly approaching the working recoveries. It is, however, emphasized that the greatest care should be taken in maintaining all the tools, and particularly the cutting shoe, in as nearly perfect condition as possible. Good results can never be attained by defective equipment, and if one maintains his drilling plant in efficient working order and exercises every care in operation, he is doing everything possible towards accuracy.

In estimating the results the formula given below has been adopted, but in the case of tin only after a number of exhaustive experiments and after a number of bores had been checked by sinking a pit around the boreholes. The bore has the advantages of speed and usefulness in very wet, marshy ground, but lacks the opportunity of being checked. A check demands a new bore, but not so in the case of a pit.

In calculating the value of a borehole done with a 4 in. drill, the cutting shoe of which has an external diameter of 4·75 in., the following formula may be applied :—

Where D = depth in ft.,
V = value in lb. per cu. yd.,
G = grams of SnO_2 (70% quality),

Then $V = 0{\cdot}471 \frac{G}{D}$

The theoretical figure is $V = 0{\cdot}4837 \frac{G}{D}$ but as stated above, after

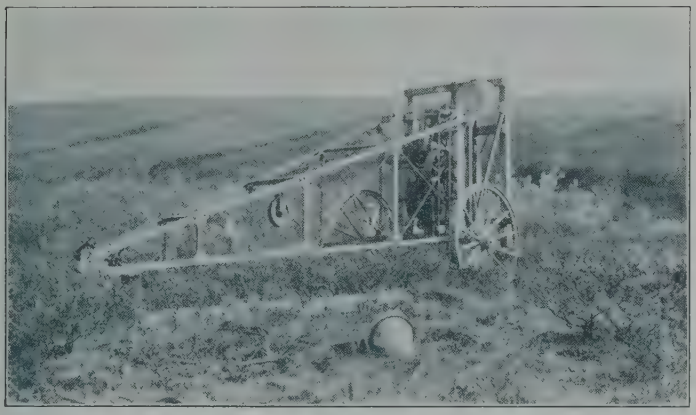

FIG. 5.—A 6 IN. CHURN DRILL READY FOR TRANSPORTING TO A NEW HOLE.

FIG. 6.—A PORTABLE 4 IN. CHURN DRILLING OUTFIT.

many tests had been run and checked by pitting around the bore the formula as first given was found to approximate to the working results most closely.

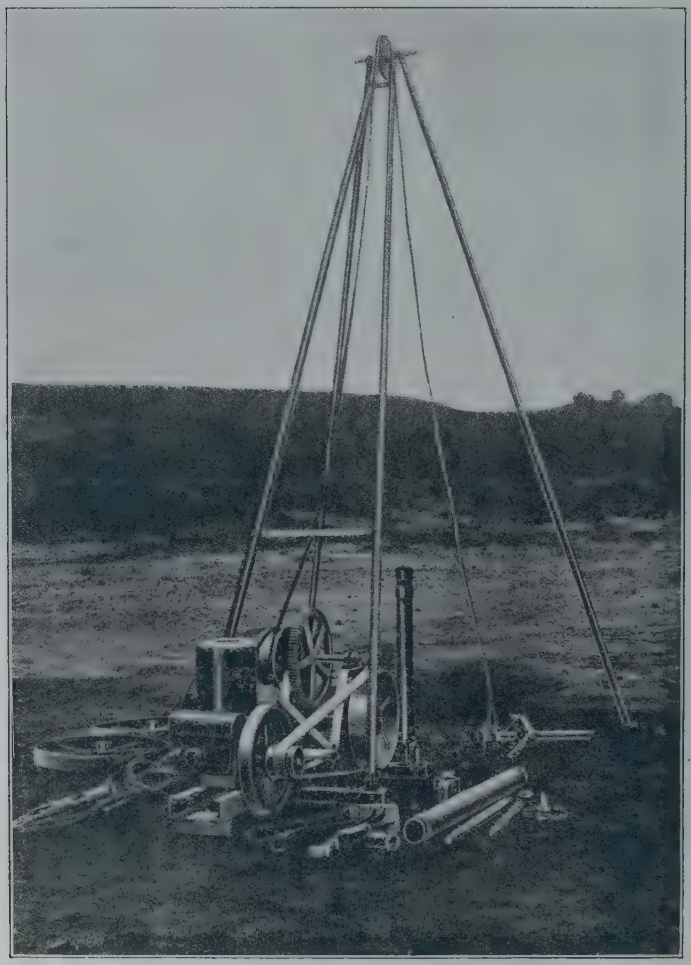

Fig. 7.—A P.A.P. 4 in. Churn Drilling Outfit.

Californian Drilling Practice and Method of Valuation.—The churn drill is the more common type used in Californian

practice. It differs from the Banka type in that the material of the core is broken and churned up, or made into a slurry sufficiently liquid to be pumped. If necessary water is added to obtain fluidity.

At the outset of operations a hole is usually sunk a foot or two in depth and the first section of pipe casing (drive pipe as it is called) with the cutting shoe attached is set up in the hole and plumbed to maintain a vertical position and driven into the ground a few feet. Clamps on the heavy drilling bit form the "monkey" for driving the pipe and a removable driving cap protects the head of the pipe from damage during the process.

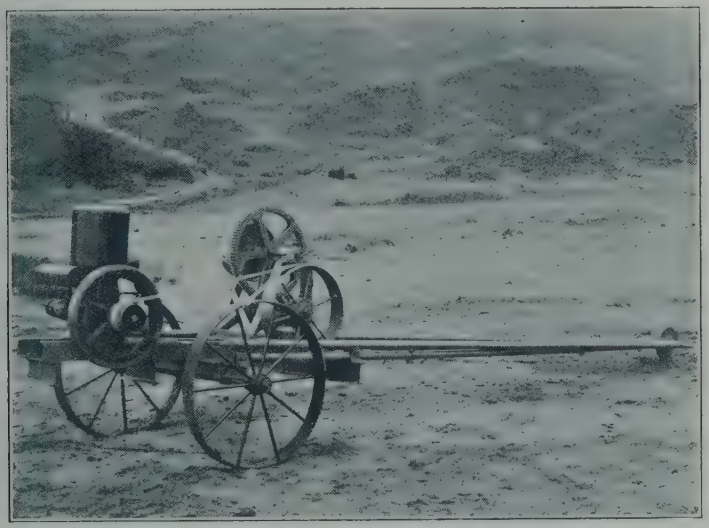

Fig. 8.—A P.A.P. 4 in. Churn Drilling Outfit Folded for Transport.

If it is known that the surface soil or overburden does not carry any values, the pipe may be driven down to gravel level, sections being added as necessary. The core in the pipe is next pumped out, but it may first be necessary to break this up with the cutting chisel. The drill is usually supplied with a small power unit and winch with a warping drum, thus enabling the operator to give a chopping action to the bit. The core is removed to within a few inches of the bottom of the pipe. This is done with the pump. In soft ground, inclined to run in, it may be advisable to retain a deeper plug of uncut material in the bottom of the pipe to act as a seal and prevent a "run in". The pipe is again

driven about a foot and the process of churning and pumping repeated and so on until bedrock is reached. It is advisable to penetrate well into the bedrock occasionally to ascertain if any values extend there. The material removed from the casing

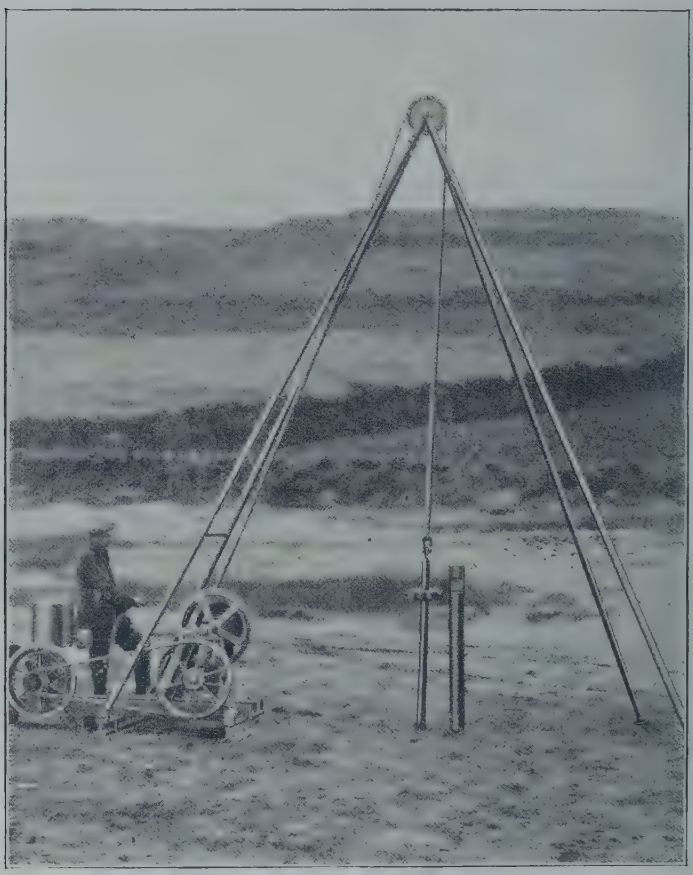

Fig. 9.—A P.A.P. 4 in. Churn Drill in Working Position.

by the pump is carefully examined and its nature and depth recorded in the field book. It is placed in a trough and carefully panned to ascertain the gold content. The volume displaced is calculated by taking the length of pipe driven to bedrock and

the external diameter of the cutting shoe. The gold recovered is measured in milligrams. (See Figs. 5–10.)

A typical field book or log is shown in Table 1. A perusal of this table will show the following details :—A drill was used with 4 in. casing which had a cutting shoe of 4·75 in. diam. externally. The area of a circle of this diameter is 17·721 sq. in., and the volume per foot of depth is 0·00455 cu. yds. (see Table). The borehole was 25 ft. deep, ∴ 25 × 0·00455 = 0·11375 cu. yds. The gold recovered from the bore amounted to 50 milligrams @ $19·50 per oz., or 0·06269 cents per m.g. (see Table 3). ∴ 50 m.g. × 0·06269 = 3·1345 cents. If we divide the value recovered in cents from the borehole by the cubic yards displaced, i.e. 0·11375, we get 27·556 cents as the value of the ground per cu. yd.

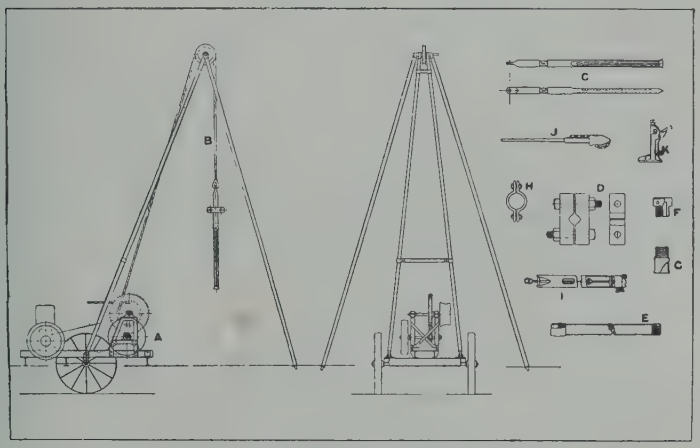

Fig. 10.—Details of a P.A.P. 4 in. Drill.

A simpler and somewhat shorter method is to use the reciprocal for the volume in cu. yds. per ft. (see Table 2) which gives these reciprocals or multiple factors for cutting shoes varying from 4 in. to 6½ in. The application of this table on the sample results above given would be as follows :—Multiply the value in cents recovered from the borehole by the multiple factor or reciprocal and divide by the depth in feet, e.g. 3·1345 × 219 = 686·455 divided by 25 = 27·458 cents.

Malayan valuations are for the most part conducted by the hand drilling method. The drills are usually smaller than 4 inches. The system of measurement is done in a box of ¼ cubic foot

capacity. The material is rammed tightly into the box so as to represent its condition *in situ* as nearly as possible. Alternatively it is measured by water displacement. It is then washed and the concentrate cleaned to a 70% tin oxide standard and weighed on a special scale so graduated that each division represents a value of 1 Kattie per cubic yard. The divisions are further sub-divided into tenths so that one-tenth of a Kattie in value can be read direct. A Kattie equals 1·33 lb. It is a notable fact, however, that in the majority of cases where dredging has been employed that the values recovered have been in excess of the prospecting valuation of the ground. This is most probably due to the fact that in ramming the material into the $\frac{1}{4}$ cubic ft. box a result is attained which only partially approaches a true representation of the volume of the material in place. See also *The Mining Magazine*, October, 1929, pp. 205–215.

TABLE 1

DRILLING LOG-BOOK.

Name of Property ..... Size of drill .... 4 in.

Locality .....

No. of borehole ..... External diam. of shoe ..... 4·75 in.

Date .....

Time of pumping.		Depth of Pipe.		Depth of Pumping.		Core. Drive.		Core. Pumping.		Core. Rem.		No. of Colours. Size.			Remarks on Formation.
Hr.	Min.	Ft.	In.	Ft.	In.	Ft.	In.	Ft.	In.	Ft.	In.	1	2	3	
7						4	9	4	7		2				Soil.
7	35	6	0	5	9	1	3	1	6		3				Fine sand.
8	15	7	0	6	9	1	0	1	5		3				Light gravel.
9	00	8	2	7	11	1	2	1	4		3			2	,, ,,
9	40	9	3	9	0	1	1	1	5		3			1	,, ,,
10	25	10	7	10	3	1	4	1	8		4			3	Small band of clay.
11	15	12	7	12	3	2	0	2	4		4		1	5	Coarse gravel.
12	00	14	9	14	5	2	2	2	8		4		4	10	,, ,,
1	30	16	9	16	6	2	0	2	4		3		1	4	,, ,,
2	10	18	10	18	6	2	1	2	5		4	1	4	8	,, ,,
2	55	20	0	19	9	2	2	2	6		3	2	8	10	,, ,,
3	30	22	0	21	9	2	0	2	4		3	1	5	5	,, ,,
4	15	25	0	25	0	2	0	2	2		0		2	4	Bedrock, soft decomposed granite.

Depth.

Soil, 4′ 9″

Gravel, 12′ 7″

Depth to bedrock, 25 ft.

Depth of pay in bedrock, 1′ 0″

Frost, nil.

Calculated volume, 0·11375 cu. yd.

Measured volume .....

Core volume .....

Wt. of gold in milligrams, 50 at $19·50 per oz.

Constant used .....

Calculated value per cu. yd. in cents, 27·556.

Time Log.

Moving, 1 hr. 35′

Drilling, 8 hr. 15′

Pulling, 2 hr. 10′

Delays, 0 hr. 35′

TABLE 2

CUBIC DISPLACEMENT AND MULTIPLE FACTORS.

Outside Diameter.	Area in Sq. Inches.	Cubic Inches per ft. length of Pipe.	Cubic Yards per Foot.	Reciprocal or Multiple Factor.	Approximately = the length of bore equivalent to 1 cu. yd. of material.
4″	12·566	150·792	·00323		309
4 1/8″	15·904	190·848	·00408		245
4 2/8″	17·721	212·652	·00455		219
5″	19·635	235·620	·00505		198
5 1/16″	20·129	241·548	·00517		193
5 1/8″	20·629	247·548	·00530		189
5 1/4″	21·648	259·776	·00554		180
5 1/2″	23·758	283·896	·00611		163
6″	28·274	339·288	·00727		137
6 1/8″	29·465	353·580	·00757		132
6 1/4″	30·680	368·160	·00789		127
6 3/8″	31·919	383·028	·00820		122
6 1/2″	33·183	398·196	·00853		117

TABLE 3

TABLE OF GOLD VALUES.

Value per Oz.	Value per Grain. Cents.	Value per Mg. Cents.
$20·67	4·306	·06646
20·00	4·166	·06430
19·50	4·062	·06269
19·00	3·985	·06103
18·50	3·854	·05948
18·00	3·750	·05787
17·50	3·646	·05626
17·00	3·542	·05466
16·50	3·437	·05305
16·00	3·333	·05144
15·50	3·229	·04983
15·00	3·125	·04823
14·50	3·021	·04662
14·00	2·917	·04500
13·50	2·812	·04340
13·00	2·708	·04180
12·50	2·604	·04019
12·00	2·500	·03858
11·50	2·396	·03697
11·00	2·297	·03537

WATER SUPPLY AND MEASUREMENT.

In alluvial mining an adequate water supply is an important factor, and in order to use it to the fullest advantage the engineer in charge should have a thorough knowledge of the laws of hydraulics. Upon him will devolve the responsibility of getting water on to his working places and whether it be by pumping, ditches, flumes, or pipelines he must be able to calculate the carrying capacity in either instance, and be able to make allowances for losses in transmission, and generally to lay out the system of supply with due regard to sound engineering practice.

The laws of gravity are the basis for the science of hydraulics, and " velocity " is a prime factor in every problem. In calculating the flow of water in pipes " velocity of efflux," together with " loss by friction ", must be considered. There are three kinds of velocity—uniform, accelerated, and retarded. It is the latter, and its cause, friction, that mining engineers are most concerned with, as velocities calculated merely from the laws of gravity do not take into consideration friction, change of direction, etc., which must be allowed for as causes in reducing the volume of water delivered through pipes, flumes, and ditches. The primary formula of $\sqrt{2gh} = V$ is the foundation of the calculations and simply means that the velocity is found by extracting the square root of the product of the head multiplied by 2 and by 32, where g = the acceleration due to gravity or 32 ft. per sec. per sec.,[1] and h = the head or height in feet. For example, there is a supply of water filling a pipe with a head of 25 feet, what is the velocity :—

$2 \times 32 \times 25 = 1{,}600$ and the sq. root of $1{,}600 = 40$. Therefore the velocity is 40 ft. per second, not taking into account the effect of friction in the pipe.

The volume discharged in cu. ft. per second (neglecting friction) is found by multiplying the cross-sectional area of the pipe (in ft.) by the velocity (in ft. per sec.).

The friction loss in a pipe varies with the velocity. If we double the velocity we increase the friction four times and if we treble it we increase the friction nine times. A head of water expressed in feet may be converted to lb. per square inch by multiplying by 0·434 which represents the weight of a column of water one square inch in area and one foot high, and conversely, pressure in lb. per square inch can be made to read in feet of head by multiplying the pressure per sq. inch by 2·304. A cubic ft. of water = 62·5 lb. = $6\frac{1}{4}$ imperial gallons = $7\frac{1}{2}$ U.S. gallons. The diameter of a pipe has no effect whatever on the pressure

[1] More accurately 32·08 to 32·25, depending on latitude.

or static head, but the size of the orifice from which the water is drawn has a very important bearing on the amount of pressure lost by friction. If the orifice and supply pipe are of equal capacity and we double the size of the pipe, we reduce the velocity three-fourths ; and the friction under such circumstances is only one-sixteenth of what it was originally. The " head of water " is the number of vertical feet from the level of supply surface (to be exact, from a point slightly below the surface) to the centre of the outlet. The theoretical velocity of efflux from an orifice in any direction is

$$V = \sqrt{2gh}$$

where h is the pressure head in feet and $g = 32{\cdot}2$.

In practice this velocity is a little less than that given by the formula and ranges from 96% to 99% of the theoretical value. In other words, this ratio of 0·96 to 0·99 is called the " coefficient of velocity ". An average usually taken is 0·97. If the velocity of efflux is known, then the head h required to produce that velocity can be found by transforming the formula, viz :— $h = \frac{V^2}{2g}$. Example :—A jet of water is issuing with a velocity of 60 ft. per second ; what is the head causing this velocity ?

$$h = \frac{V^2}{2g} = \frac{60^2}{2 \times 32{\cdot}2} = 55{\cdot}9 \text{ ft.} \quad \text{Answer.}$$

Co-Efficient of Contraction.—When water flows through an orifice, see Fig. 11, the direction of flow at the sides causes the vein of water to contract as it issues to an extent that it only equals from 60 to 70% of the full area of the orifice, the actual amount of contraction depending on the size of the orifice and the pressure. It has already been shown that the discharge of any orifice pipe or channel is equal to the cross-sectional area of the stream in square feet multiplied by its velocity in feet and this gives the discharge in cubic feet per second and that the " co-efficient of velocity " varies from 0·96 to 0·99 (usually taken at 0·97). If we take the " co-efficient of contraction " as 0·65, we then have the following formula for discharge :—

$$Q = 0{\cdot}97\ v \times 0{\cdot}65\ A.$$

Where Q = discharge in cubic feet per second.
v = theoretical velocity of efflux in feet per second.
A = area of orifice in square feet.

substituting v for its value $\sqrt{2gh}$ we have the equation

$$Q = 0{\cdot}97 \times 0{\cdot}65\ A\ \sqrt{2gh}.$$
$$= 0{\cdot}63\ A\ \sqrt{2gh}.$$

The co-efficient 0·63 is known as the " co-efficient of discharge ", which, as already shown, will vary with different conditions. It

is, therefore, desirable to substitute "c" for the co-efficient of discharge, and the formula then becomes

$$Q = c\ A\ \sqrt{2gh}.$$

The "co-efficient of discharge" is the product of the "co-efficient of velocity" and the "co-efficient of contraction" and its value for circular orifices may be taken as varying from 0·63 for low heads to 0·59 for higher heads.

Friction of Water in Pipes.—Hydraulic operators are often inclined to view the engineer as a crank and faddist when he emphasizes the necessity of careful design in the pipeline so as

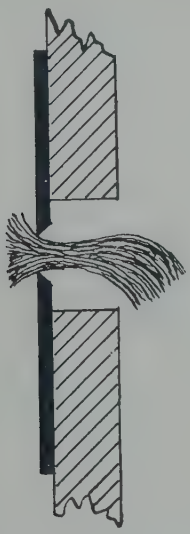

FIG. 11.—ILLUSTRATING CONTRACTION OF ORIFICE.

to avoid friction losses in operating. They view statements of losses with doubt, and rarely, if ever, endeavour to carry out the simple test of using a pressure gauge near the point of discharge—reading it first when the pipe is standing full of water and then again while the water is discharging. The difference between the two readings show the relation between the hydrostatic pressure and the hydraulic head.

A wealth of experimental research has been made in ascertaining the amount of friction there is when water is flowing through pipes. The various results are of academic rather than practical interest to the mine operator, but it has been shown

that the friction loss in a pipe is approximately proportional to its length and to the square of the velocity of the water, and is inversely proportional to the cross-section of the pipe divided by its circumference.

The important point is that the mean average velocity should not exceed 3·5 feet per second. When this velocity is exceeded, the friction losses increase very rapidly with the consequent result of a large loss of head. The friction primarily depends on the velocity of flow through the pipes and the relation that the sectional area of the pipe bears to its perimeter. The condition of the internal surface also has a bearing on the problem. The friction in small pipes is greater in proportion to the velocity of the water than in large ones.

For the purpose of calculating the loss of head due to friction, the authors consider that the following formula gives as effective results as any put forth :—

$L = \frac{p\,l\,c\,v^2}{2ga}$; where

L = the loss of head in 100 ft. length of pipe due to velocity of water ;
p = circumference of pipe in feet ;
l = 100 ft. ;
c = a co-efficient varying from 0·00406 to 0·01338, according to diameter of pipe and velocity of water ;
v = the velocity of water in feet per second ;
g = the acceleration due to gravity, 32·2 ft. per sec. per sec. ;
a = the cross-sectional area of pipe, in feet ;

Table No. 4 is calculated from this formula, and the following two examples will illustrate its application :—

Example No. 1.

There is in use an old riveted pipeline 12 in. diameter 2,000 ft. long, the difference between entry and outlet being 200 ft., and it is carrying 8 cubic feet per second. Required the total loss of head and operating pressure at the point of delivery. The area of a 12 in. pipe = 0·785 sq. feet, and $\frac{8}{0{\cdot}785} = 10{\cdot}19$, the velocity in ft. per sec. By referring to the table, under the head of 12 in. pipe the co-efficient for 10 feet per second is 6·308, and for 11 feet per second is 7·570, and 7·570 − 6·309 = 1·262, and 1·262 × 0·19 = 0·340. This must be added to the co-efficient for a velocity of 10 ft. per second, viz. 6·308 + 0·340 = 6·648. The total length of pipe is 2,000 feet, and as the loss of head in each 100 ft. = 6·648 ft., this multiplied by 20 = 132·96 ft., which is the total loss of head. Put in other words the effective pressure is 57·70 lb.

TABLE 5.

TABLE SHOWING THE NUMBER OF CUBIC FEET OF WATER DISCHARGED PER SECOND IN CLAIM PIPES (Eytelwein's Formula, $Q = \sqrt{R\,S \times 10{,}560} \times a$), calculated by Mr. Alexander Aitken, Manager of Government Water-races at Kumara, on the West Coast of Middle Island, New Zealand.

Total Length divided by Total Fall of Hydraulic Inclination.	Diameter of Pipes in Inches.																					
	6	7	8	9	10	11	12	13	14	15	16	17	18	20	22	24	26	28	30	32	34	36
1 in 10	2·255	3·516	4·631	6·217	8·090	10·265	12·761	15·587	18·761	22·293	26·190	30·480	35·166	45·760	58·070	72·189	88·180	106·130	126·110	148·190	172·440	198·930
1 in 20	1·595	2·345	3·274	4·397	5·720	7·259	9·023	11·022	13·266	15·763	18·523	21·554	24·866	32·395	41·065	51·044	62·352	75·042	89·170	104·880	121·930	140·660
1 in 30	1·304	1·916	2·677	3·593	4·676	5·930	7·377	9·011	10·846	12·897	15·144	17·623	20·330	26·456	44·573	41·732	50·977	61·353	72·903	85·668	99·687	115·000
1 in 40	1·127	1·657	2·315	3·108	4·045	5·132	6·380	7·798	9·385	11·146	13·095	15·240	17·588	22·880	29·035	36·094	44·090	53·065	63·055	74·095	86·220	99·465
1 in 50	1·088	1·483	2·071	2·780	3·617	4·591	5·706	6·971	8·390	9·969	11·715	13·632	15·726	20·466	25·972	32·283	39·435	47·462	56·397	66·271	77·117	88·962
1 in 60	0·920	1·354	1·890	2·537	3·302	4·191	5·209	6·363	7·659	9·100	10·691	12·445	14·356	18·640	23·709	29·470	35·999	43·327	51·483	60·497	70·285	81·211
1 in 70	0·852	1·253	1·750	2·349	3·057	3·880	4·823	5·891	7·090	8·425	9·900	11·521	13·291	17·296	21·950	27·284	33·328	40·111	47·663	56·008	65·174	75·185
1 in 80	0·797	1·172	1·637	2·198	2·860	3·629	4·511	5·511	6·632	7·881	9·261	10·777	12·433	16·196	20·532	25·522	31·176	37·521	44·585	52·440	60·960	70·330
1 in 90	0·751	1 108	1·543	2·072	2·696	3·422	4·253	5·195	6·253	7·430	8·731	10·184	11·721	15·254	19·403	24·062	29·392	35·375	42·035	49·395	57·478	66·307
1 in 100	0·713	1·048	1·464	1·965	2·558	3·246	4·035	4·929	5·932	7·049	8·283	9·639	11·110	14·471	18·365	22·828	27·885	33·560	39·878	46·861	54·529	62·905
1 in 120	0·652	0·958	1·338	1·796	2·338	2·967	3·688	4·005	5·423	6·448	7·572	8·811	10·165	13·228	16·786	20·866	25·488	30·676	36·451	42·834	49·843	57·500
1 in 160	0·563	0·828	1·157	1·554	2·022	2·566	3·190	3·899	4·692	5·573	6·547	7·620	8·799	11·440	14·517	18·047	22·045	26·537	31·527	37·092	43·110	49·732
1 in 200	0·504	0·741	1·035	1·390	1·808	2·295	2·853	3·485	4·195	4·984	5·857	6·816	7·863	10·233	12·986	16·141	19·767	23·731	28·188	33·135	38·558	44·480
1 in 240	0·460	0·677	0·945	1·268	1·651	2·095	2·604	3·181	3·829	4·500	5·345	6·222	7·178	9·320	11·854	14·735	17·999	21·663	25·741	30·248	35·117	40·605
1 in 280	0·426	0·626	0·875	1·174	1·528	1·940	2·411	2·945	3·545	4·212	4·950	5·765	6·645	8·648	10·975	13·642	16·664	20·055	23·831	28·004	32·587	37·592
1 in 320	0·398	0·586	0·818	1·099	1·430	1·814	2·255	2·755	3·316	3·945	4·630	5·388	6·216	8·098	10·266	12·761	15·588	18·760	22·296	26·220	30·480	35·160
1 in 360	0·375	0·554	0·771	1·036	1·348	1·711	2·126	2·597	3·126	3·715	4·365	5·092	5·860	7·627	9·701	12·031	14·696	17·687	21·017	24·697	28·739	33·153
1 in 400	0·356	0·524	0·732	0·982	1·279	1·623	2·017	2·464	2·966	3·524	4·141	4·819	5·560	7·235	9·182	11·414	13·942	16·780	19·939	23·430	27·264	31·452
1 in 480	0·326	0·479	0·669	0·898	1·169	1·483	1·844	2·202	2·711	3·224	3·786	4·405	5·083	6·614	8·393	10·433	12·744	15·338	18·225	21·417	24·921	28·750
1 in 640	0·281	0·414	0·578	0·777	1·011	1·283	1·585	1·949	2·346	2·781	3·273	3·820	4·399	5·720	7·258	9·023	11·027	13·268	15·763	18·521	21·555	24·866
1 in 800	0·252	0·370	0·517	0·695	0·904	1·147	1·426	1·743	2·097	2·492	2·928	3·404	3·931	5·116	6·493	8·070	9·883	11·865	14·099	16·567	19·279	22·240
1 in 960	0·230	0·338	0·472	0·634	0·825	1·047	1·302	1·590	1·914	2·275	2·672	3·111	3·589	4·660	5·927	7·367	8·999	10·831	12·870	15·124	17·558	20·302
1 in 1120	0·213	0·313	0·437	0·587	0·764	0·970	1·205	1·472	1·772	2·106	2·475	2·882	3·322	4·324	5·487	6·821	8·332	10·027	11·915	14·002	16·293	18·796
1 in 1280	0·199	0·293	0·409	0·549	0·715	0·907	1·127	1·377	1·658	1·970	2·315	2·694	3·108	4·049	5·133	6·380	7·794	9·380	11·146	13·110	15·240	17·580
1 in 1440	0·187	0·277	0·385	0·518	0·674	0·855	1·063	1·298	1·563	1·857	2·182	2·546	2·930	3·813	4·850	6·015	7·348	8·843	10·508	12·348	14·369	16·576
1 in 1600	0·178	0·262	0·366	0·491	0·639	0·811	1·058	1·232	1·483	1·762	2·070	2·409	2·780	3·617	4·591	5·707	6·971	8·391	9·969	11·715	13·632	15·726
1 in 1920	0·163	0·289	0·334	0·449	0·584	0·741	0·972	1·001	1·355	1·612	1·892	2·202	2·541	3·307	4·196	5·216	6·372	7·669	9·912	10·708	12·460	14·370
1 in 2560	0·140	0·207	0·289	0·388	0·505	0·641	0·797	0·974	1·173	1·390	1·636	1·405	2·199	2·860	3·619	4·511	5·513	6·634	7·881	9·265	10·777	12·433
1 in 3200	0·126	0·185	0·258	0·347	0·452	0·573	0·713	0·871	1·048	1·246	1·464	1·702	1·965	2·558	3·246	4·035	4·941	5·933	7·049	8·783	9·639	11·120

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TABLE 6.

Table showing the Quantity of Water that Pipes will Discharge in Cubic Feet per Second, Calculated from Neville's Formula.

$$V = 140\sqrt{RS} - 11\sqrt[3]{RS}.$$

Diameters of Pipes in Inches.

	7	8	9	10	11	12	13	13½	14¾	15	15½	16½	17½	18	18½	19	20	21	22	24	26	28	30
1 in 10	3·80	5·32	7·17	9·37	11·91	14·86	18·20	19·70	25·02	26·12	28·43	33·23	37·56	41·40	44·40	47·46	53·98	60·67	68·71	85·60	104·75	126·26	149·75
1 in 20	3·45	3·88	4·96	6·48	8·25	10·29	12·79	13·71	17·34	18·10	19·70	23·04	26·77	28·59	30·78	32·92	37·48	42·40	47·68	59·20	72·76	87·71	104·39
1 in 30	2·11	2·99	4·00	5·21	6·64	8·28	10·73	11·05	13·97	14·59	15·88	18·57	21·55	23·14	25·17	26·54	30·22	34·20	38·47	47·94	58·71	70·82	85·33
1 in 40	1·80	2·53	3·36	4·49	5·39	7·10	8·70	9·47	11·98	12·50	13·61	15·83	18·49	19·85	21·26	22·77	25·93	29·34	33·06	41·15	50·59	61·79	72·40
1 in 50	1·59	2·24	3·00	3·96	5·04	6·29	7·12	8·40	10·62	11·10	12·10	14·30	16·40	17·61	18·89	20·20	23·00	26·00	29·20	36·31	44·00	53·98	61·00
1 in 60	1·42	2·01	2·70	3·56	4·40	5·50	6·82	7·50	9·08	9·90	10·55	12·23	14·80	15·50	16·90	18·32	20·30	23·08	26·39	31·50	38·26	47·58	53·56
1 in 70	1·31	1·86	2·50	3·27	4·11	5·10	6·39	6·90	8·50	8·94	9·88	11·41	13·66	14·42	15·80	17·00	20·00	21·24	23·99	29·30	35·44	44·08	49·97
1 in 80	1·20	1·72	2·28	3·00	3·80	4·70	5·91	6·37	7·91	8·35	9·22	10·63	12·53	13·40	14·80	15·00	17·83	19·84	22·91	27·54	32·89	40·26	45·57
1 in 90	1·13	1·60	2·13	2·89	3·55	4·38	5·45	5·93	7·38	7·77	8·59	9·91	11·50	12·51	14·03	14·86	16·36	18·43	20·68	25·91	30·72	37·77	42·77
1 in 100	1·07	1·51	2·02	2·67	3·29	4·12	5·15	5·58	6·93	7·37	8·08	9·31	10·97	11·77	12·98	13·94	15·15	17·23	19·35	24·34	28·80	35·91	40·37
1 in 125	0·93	1·31	1·76	2·32	2·86	3·61	4·45	4·88	6·09	6·44	7·19	8·17	9·63	10·30	11·38	11·84	13·05	15·23	17·16	21·20	25·19	30·64	34·77
1 in 150	0·84	1·20	1·54	2·10	2·46	3·28	4·07	4·38	5·46	5·73	6·36	7·34	8·65	9·27	10·23	10·60	12·00	13·62	15·39	19·10	22·73	27·79	31·58
1 in 175	0·75	1·08	1·43	1·91	2·34	2·96	3·69	4·02	5·00	5·31	5·82	6·71	7·91	8·39	9·20	9·65	11·18	12·42	14·08	17·54	20·58	25·65	29·18
1 in 200	0·68	0·99	1·32	1·77	2·22	2·77	3·38	3·70	4·62	4·90	5·38	6·21	7·32	7·80	8·65	8·98	10·18	11·42	12·98	16·23	19·05	23·87	27·18
1 in 250	0·62	0·87	1·21	1·54	1·90	2·38	2·99	3·29	4·09	4·40	4·83	5·56	6·44	6·92	7·61	8·00	9·09	10·22	11·44	14·13	17·25	20·67	23·58
1 in 300	0·56	0·78	1·10	1·36	1·70	2·12	2·69	3·10	3·67	3·88	4·36	5·02	5·80	6·18	6·86	7·38	8·18	9·02	10·12	12·79	15·45	18·89	21·58
1 in 350	0·54	0·72	0·99	1·27	1·59	2·00	2·46	2·72	3·39	3·48	3·91	4·59	5·31	5·71	6·27	6·74	7·45	8·21	9·46	11·78	14·40	17·46	19·99
1 in 400	0·52	0·67	0·90	1·19	1·51	1·89	2·30	2·56	3·19	3·27	3·59	4·27	4·92	5·30	5·82	6·12	6·90	7·61	8·68	10·96	13·60	16·13	18·98
1 in 440	0·45	0·64	0·86	1·13	1·44	1·79	2·19	2·42	3·03	3·15	3·43	3·95	4·66	5·00	5·37	5·72	6·53	7·38	8·28	10·34	12·65	15·26	18·15
1 in 480	0·43	0·61	0·82	1·08	1·37	1·70	2·09	2·30	2·86	3·00	3·27	3·76	4·44	4·77	5·11	5·46	6·21	7·02	7·90	9·84	12·04	14·52	17·21
1 in 528	0·41	0·58	0·78	1·02	1·30	1·61	1·98	2·19	2·70	2·85	3·09	3·57	4·20	4·52	4·84	5·17	5·89	6·66	7·49	9·33	11·41	13·76	16·35
1 in 587	0·39	0·55	0·73	0·96	1·22	1·52	1·88	2·06	2·55	2·68	2·93	3·37	3·95	4·25	4·56	4·87	5·55	6·28	7·06	8·79	10·79	13·00	15·36
1 in 660	0·36	0·51	0·69	0·90	1·15	1·43	1·77	1·93	2·38	2·51	2·73	3·15	3·71	3·98	4·26	4·56	5·20	5·88	6·60	8·24	10·06	12·14	14·42
1 in 754	0·34	0·48	0·64	0·84	1·06	1·33	1·63	1·79	2·22	2·33	2·53	2·92	3·44	3·69	3·96	4·23	4·82	5·45	6·13	7·68	9·33	11·26	13·38
1 in 880	0·31	0·43	0·58	0·76	0·97	1·21	1·48	1·64	2·04	2·13	2·32	2·72	3·16	3·39	3·63	3·88	4·42	4·99	5·62	7·08	8·56	10·33	12·46

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per sq. in. less than that due to the hydrostatic head, e.g. 2,000 × 0·434 lb. = 868 lb. per sq. in. and 132·96 × 0·434 = 57·7 ∴ the effective pressure is 868 – 57·7 or 810·3 lb. per square inch.

Example No. 2.

Taking a 15 in. pipe operating under the same conditions, the figures would be as follows :—

The loss of head in 100 ft. of pipe due to a

Velocity of 7 ft. per sec.	=	2·444
,, 6 ft. per sec.	=	1·834
Difference	=	0·610

$$\text{The Velocity is } \frac{8}{1{\cdot}227} = 6{\cdot}52 \text{ ft. per sec.}$$

$$\therefore 0{\cdot}610 \times 6{\cdot}52 = 0{\cdot}317.$$

The loss due to a velocity of 6 ft. per sec.....	=	1·834
The loss due to a velocity of 6·52 ft. per sec...	=	0·317
The total loss per 100 ft	=	2·151 ft.

Therefore 2·151 ft. × 20 = 43·02 ft. The loss of head is 18·67 lb. per sq. in. And the hydrostatic head is 2,000 × 0·434 = 868 and 868 – 18·67 = 849·33 lb. per sq. in.

Measurement of Water over Weirs and Notches.[1]—The laws relating to the flow of water in this manner have been studied and investigated, both mathematically and experimentally, for many years, and are well known. The names of Hawksley, Rankine, Bazin, Jas. Thomson, Francis, Unwin, and many others are associated with this subject, and there is no doubt it is one of great interest and utility.

Rectangular Weirs.

Theoretical Considerations.—Looking at this matter in its simplest and most fundamental form, it is obvious that the quantity of water which will flow, under gravity, through a rectangular weir, is the product of the cross-sectional area of the flow and the average velocity, in other words that :—

Cubic feet per second $= Q$ = area in square feet × average velocity in feet per second.

At first sight therefore, it would appear that the problem of measurement should be a fairly simple and straightforward matter, and so it is if only rough approximations are required.

[1] Supplied by the courtesy of The Lea Recorder Co., Ltd., 28, Deansgate, Manchester.

e.g.—If l = width of weir in feet, and H = head on weir, in feet,
Then $l \times H$ = cross-sectional area.

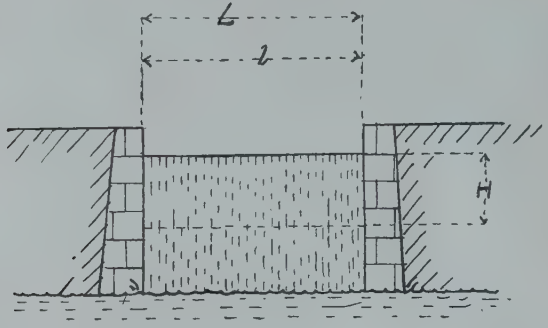

Fig. 12.—Illustrating Contraction of Flow over Weir.

Seeing also that the only force causing the water to flow at all is gravity, it is fairly obvious that the average velocity must depend, more or less, upon the laws of falling bodies, viz. upon $\sqrt{H}$.

Hence, therefore, we should expect the formula for the flow to be of the form :—

$$Q = lH \times \sqrt{H} \times K$$
$$= lH\sqrt{H} \times K = lH^{\frac{3}{2}} \times K$$

It is so, and this is the formula for all such flows.

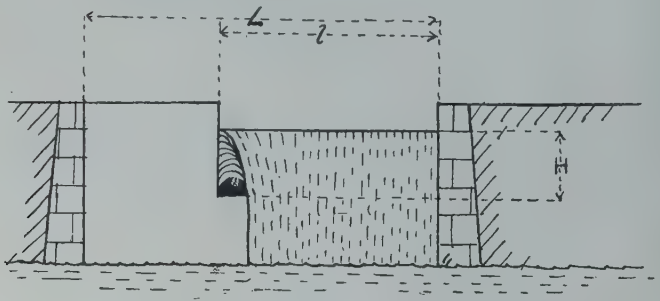

Fig. 13.—Illustrating Contraction of Flow over Weir.

When, however, we begin to investigate more closely as to *what is the exact area* and *what is the exact average velocity*, the problem becomes more complicated, the principal cause of the

complication arising from the fact that the water does not flow in a parallel horizontal manner through the weir, but along curved and converging stream lines, lines which converge both inwards and upwards, depending upon the width of the weir as compared with the width of the stream, and also upon the depth of the bed of the stream below the edge of the weir. Figs. 12, 13 and 14 illustrate this point, viz., the contractions of the flow.

In Fig. 12, where the weir is of the same width as the channel, i.e. where $l = L$, there are no end contractions, but there is of course a contraction at the crest or edge of the weir, due to the water springing upwards from the bottom of the channel, as seen in the figures mentioned.

In Fig. 13, where one side of the weir coincides with the side of the channel, there is only one end contraction.

In Fig. 14, where the weir is placed more or less centrally in the channel, there are two end contractions.

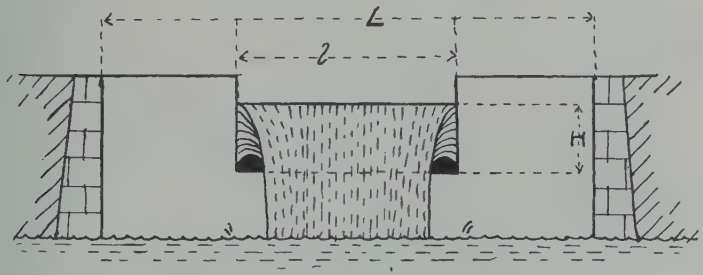

FIG. 14.—ILLUSTRATING CONTRACTION OF FLOW OVER WEIR.

For such cases Rankine gives the following formula :—

If Q = cubic feet per second, H = Head in feet, l = Length of weir in feet, and

c = a constant depending upon the ratio of the width of weir to width of channel, i.e. upon the ratio of l to L,

Then $Q = 5{\cdot}35\ clH^{\frac{3}{2}}$, the value of c varying from 0·67 in the case of Fig. 12 to 0·595 where the stream is four times as wide as the weir.

Francis' Formula.—As this formula is very well-known, and has been in most general use for weirs with full end contractions, a few remarks deduced from Professor Unwin's writings might be quoted :—

> Mr. J. B. Francis' experiments shewed that a perfect end contraction (when the heads varied from 3 in. to 24 in. and the lengths of the weirs were not less than three times the head) diminished the effective length of the weir by an

amount approximately equal to 0·10 of the head, and that therefore, if l = actual length of the notch or weir, and H = the head, then the effective width of an ordinary weir, with two end contractions, would be $= l - 0{\cdot}2H$.

Hence his general formula : $Q = 5{\cdot}35c.\ (l - 0{\cdot}1nH)\ H^{\frac{3}{2}}$ where n equals the number of end contractions. Francis found for c the mean value of 0·622, the weir being sharp edged.

Substituting the figure 0·622 for c in the above equation, we get : $Q = 3{\cdot}33\ (l - 0{\cdot}1nH)\ H^{\frac{3}{2}}$, which is the Francis formula, as most commonly used.

Recent Investigations.—Owing, however, to the great and increasing practical importance of this subject in connection with problems dealing with domestic water supply, supplies for mining purposes, sewage measurement, and numberless other industrial and power station purposes, this subject has been still further probed into, analysed, and investigated in recent years, and Mr. A. A. Barnes, of Manchester, has given us two formulæ which he claims to be more scientifically accurate and of more general and universal application. His formulæ are as follows :—

(1) For weirs with end contractions (or Francis' weirs) $Q = 3{\cdot}324H^{1{\cdot}49}\ l^{1{\cdot}11}\ (l + 2H)^{0{\cdot}11}$ and

(2) For weirs without end contractions (or Bazin weirs) $Q = 3{\cdot}324H^{1{\cdot}49}\ l^{0{\cdot}98}\ (l + 2H)^{0{\cdot}02}$.

(See " Hydraulic Flow Reviewed " by A. A. Barnes).

In view, therefore, of the elaborate and painstaking researches which have extended over the past fifty years, it may safely be said that it is now possible to measure large volumes of water by means of rectangular weirs in the most accurate manner, provided, of course, that the circumstances of each case are properly and scientifically considered, and that suitable instruments are available for interpreting the formulæ employed.

The V Notch.—When we come to the V Notch, we meet with a unique and almost wonderful example of the accuracy of weir measurement. For measuring comparatively small quantities of water, such as the dry weather flow of streams, the measurement of sewage, the discharges from pumps, etc., and especially where the flow is subject to great fluctuation, there is nothing so simple, so convenient, or so extraordinarily accurate as the V Notch, and this fact is now universally recognized. For the earliest researches and data on this subject we have to thank Professor Jas. Thomson. His experiments were made about sixty years ago at Carr's Glen, near Belfast, where a small mill pond was available for his purpose, and he used barrels or tubs for measuring the water discharged, at various rates of flow, through 90° V Notches. The account of his experiments is very interesting, especially where he refers to the small sums of money, viz. £10 and £20 voted to him at different times by the British Association for prosecuting his

researches. Although the apparatus at his disposal was of a very simple and crude nature in comparison to what is now available, his experiments were most accurately made, and recent investigations have fully confirmed his results, within small fractions, and incidentally, his great skill as an experimenter, thus showing that it is the man, and not the appliances, which is the principal factor in all original research.

Quoting his words, after making very extended experiments, he says :—

> " The results of my experiments are highly satisfactory . . . I am confident of them being very accurate . . . I find them to be in close accordance with the law which had been indicated by theoretical considerations."

He argued thus :—

(1) That the rate of flow at any instant should be proportional to the cross-sectional area of the section, and therefore to H^2.

(2) That the rate of flow at any instant should be proportional to $\sqrt{H}$, from considerations as to velocity.

He therefore concluded that the formula for expressing the flow at any time would be of the form :—

$$Q = H^2 \times \sqrt{H} \times \text{constant},$$

and this he proved to be the case in a most conclusive manner, the value of the constant being :—

2·536 where Q = cubic feet per sec. and H = head in feet,
or 0·305 where q = cubic feet per min. and h = head in inches.

(See *Proceedings of British Association* for 1858 and 1861.)

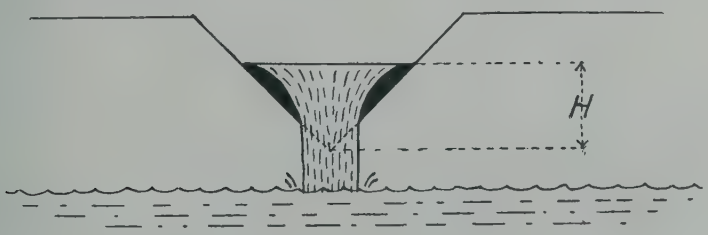

Fig. 15.—Illustrating Contraction of Flow over 90° V Notch.

In 1908, a research was instituted at Glasgow University by Professor Barr, with the object of further investigating and checking the results obtained by Dr. Jas. Thomson. The most accurate apparatus was used in this research, and every possible means was taken to eliminate errors. The experiments were also extended, and flows up to 11 in. deep were measured. The average

constant given by Thomson, viz. 0·305, was found to be correct with flows about 4 in. deep, but with a small flow of 2 in. the constant was found to be 0·3104, and for a larger flow of 10 in. it was 0·2995, the maximum difference being 0·0109.

Mr. A. A. Barnes, to whom we have previously referred in connection with rectangular weirs, has, however, discovered that by reducing the index in the V Notch formula from 2·5 to 2·48 it is possible to obtain a *real and unvarying constant* or multiplier in the equation, and his formula is as follows :—

$$Q = 2{\cdot}48 \times H^{2{\cdot}48}$$

This is really a remarkable formula, not only on account of its simplicity but on account of the strange coincidence of the constant and index being exactly alike.

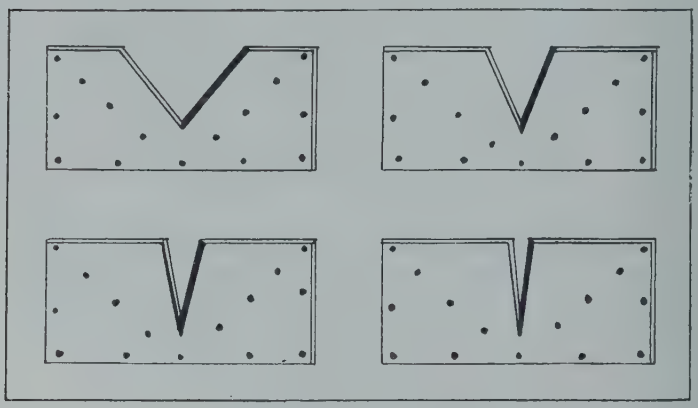

Fig. 15a.—Types of Acute Angle V Notches.

Use of Acute-Angled Notches.—For dealing with very small quantities of water or other liquids The Lea Recorder Co., Ltd., have for a number of years made use of notches of a less angle than 90 degrees, which they have termed $\frac{1}{2}$ 90°, $\frac{1}{4}$ 90°, and so on, as shown in the accompanying illustration, Fig. 15a. By means of such notches it is possible to get a larger float movement, and consequently a higher degree of sensitiveness and accuracy, than it would be with an ordinary 90° V. A series or row of notches may also be used in special cases.

Gauging of Streams.—Should the volume of discharge of a stream or river be so large as to preclude the use of a temporary weir, recourse must be had to other means of measurement. That of

TABLE 7.

Tables of Flows for V Notches and Rectangular Weirs.

(1) For 90° V Notches.

Deduced from Thomson's Formula : $Q = 0{\cdot}305\ H^{\frac{5}{2}}$.

Where Q = cubic ft. per minute, and H = head in inches.

Depth of flow in inches.	Discharge.			Depth of flow in inches.	Discharge.		
	Gallons per min.	Gallons per hour.	Gallons per 24 hours.		Gallons per min.	Gallons per hour.	Gallons per 24 hours.
1″	1·9	114	2,736	9″	463·3	27,796	667,100
2″	10·8	648	15,552	10″	602·9	36,174	868,180
3″	29·72	1,783	42,792	11″	765·1	45,903	1,101,700
4″	61·02	3,661	87,864	12″	947·9	56,872	1,364,900
5″	106·6	6,394	153,460	13″	1,158	69,471	1,667,300
6″	168·1	10,086	242,060	14″	1,394	83,611	2,006,700
7″	247·2	14,829	355,990	15″	1,656	99,351	2,384,400
8″	345·1	20,706	496,940	16″	1,946	116,745	2,801,885

N.B.—1 Gallon = 10 lb.

(2) For Rectangular Weirs.
(Without end contractions) per foot width of Weir.

Deduced from Francis' Formula, $Q = 3{\cdot}33\ lH^{\frac{3}{2}}$.

Where Q = cubic ft. per sec., l = width of Weir in ft., H = head in ft.

Depth of flow in inches	Discharge per ft. of width.				Depth of flow in inches	Discharge per ft. of width.			
	Cubic ft. per sec. = Q.	Cubic ft. per hour.	Gallons per hour.	Gallons per 24 hours.		Cubic ft. per sec. = Q.	Cubic ft. per hour.	Gallons per hour.	Gallons per 24 hours.
1″	0·080	228·0	1,795	43,080	10″	2·533	9,118·8	56,800	1,363,200
2″	0·227	817·2	5,091	122,000	11″	2·922	10,519	65,540	1,573,000
3″	0·416	1,497·6	9,333	224,000	12″	3·330	11,958	74,680	1,792,300
4″	0·641	2,307·6	14,370	344,880	14″	4·196	15,106	94,200	2,260,800
5″	0·896	3,225·6	20,080	481,920	16″	5·126	18,454	115,000	2,760,000
6″	1·117	4,237·2	26,400	633,600	18″	6·118	22,025	137,000	3,288,000
7″	1·484	5,342·4	33,280	798,720	20″	7·163	25,787	160,300	3,847,200
8″	1·812	6,523·2	40,640	975,360	24″	9·419	33,908	211,300	5,070,000
9″	2·163	7,786·8	48,500	1,164,000	30″	13·163	47,387	295,200	7,085,000

taking cross-sections of the flowing stream will give reasonably reliable data and the method of procedure is as follows :—

A site is selected where there is an even regular flow of water and four of five cross-sections are carefully taken being marked off along the bank at equal distances from one another and the depth readings taken at regular distances and recorded after the manner of the accompanying sketch Fig. 16. The average depth would therefore be :—

$$\frac{0 + 1{\cdot}2 + 2{\cdot}2 + 3{\cdot}0 + 3{\cdot}1 + 3{\cdot}2 + 3{\cdot}6 + 3{\cdot}7 + 3{\cdot}6 + 3{\cdot}3 + 3{\cdot}1 + 2 + 0}{12}$$

$$= \frac{32}{12} = 2{\cdot}66 \text{ ft.}$$

If the width is 36 feet over all the section will be $36 \times 2{\cdot}66 = 95{\cdot}76$. The average of four or five sections will therefore be the average area of water moving forward between the sections.

The next step to complete the calculation will be to determine the velocity of flow of the stream. This is done by using a float

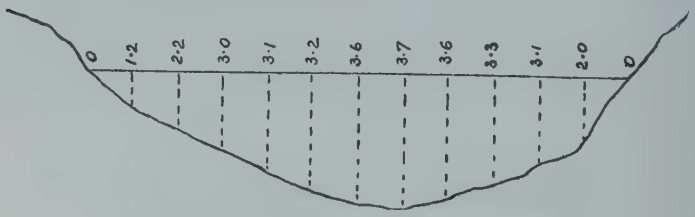

Fig. 16.—Showing Method of Gauging Streams.

and recording the time taken in passing from the first to the last cross-section. A number of observations should be made, the mean of which will give the average *surface* velocity of the stream. The mean velocity of flow will, however, be less than this, due of course to the frictional resistance of the bed and sides of the channel in which the water flows. It will vary according to the nature of the ground but may be taken approximately as equal to 0·75 of the mean surface velocity, which we may as an example assume to be 4·5 feet per second ; then $4{\cdot}5 \times 0{\cdot}75 = 3{\cdot}37$, the mean velocity of flow. Therefore the average cross-sectional area of $95{\cdot}76 \times 3{\cdot}37 = 322{\cdot}71$ cubic feet per second, the quantity of water that is being discharged by the stream.

A close approximation may be made by using a current meter and recording accurately the flow of the stream at various depths below the surface, but the method above given will meet the requirement of most mining engineers without the necessity of entering into more exact hydraulic computations.

The following table will be of service in calculating the quantity of water received per acre and give some idea of the run-off to be expected from a catchment area :—

TABLE 8.

Inches of Rainfall.	Catchment per acre in cu. ft.	Catchment per sq. mile in millions of cu. ft.
1	3,630	$2\frac{1}{3}$
2	7,260	$4\frac{1}{2}$
3	10,890	7
4	14,520	$9\frac{1}{4}$
5	18,150	$11\frac{1}{2}$
6	21,780	14
7	25,410	$16\frac{1}{4}$
8	29,040	$18\frac{1}{2}$
9	32,760	21
10	36,300	$23\frac{1}{4}$
12	43,560	$27\frac{3}{4}$
15	54,450	$34\frac{3}{4}$
20	72,600	$46\frac{1}{2}$
25	90,750	58
30	108,900	$69\frac{3}{4}$
40	145,200	93
50	181,500	$116\frac{1}{4}$
60	217,800	$139\frac{1}{2}$
70	254,100	$162\frac{1}{2}$
80	290,400	$185\frac{3}{4}$
90	326,700	209
100	363,000	$232\frac{1}{4}$

Miner's Inch and Sluice-Head.—In dealing with water measurements students are advised to shun such measurements as the "Miner's Inch" and the "Sluice-Head". The "Miner's Inch" is a variable quantity according to the country of its adoption. In California 40 miner's inches = a flow of 1 cu. ft. per sec., in Colombia it is 38·4 and other American States accept 50 "inches" as equivalent to 1 cu. ft. per sec. In the case of the "Sluice-Head", one such sluice-head is generally accepted as being equal to a flow of 1 cu. ft. per sec. This is so in New Zealand, Australia and the United States of America, but in Tasmania, where much alluvial mining is carried on, a sluice-head by statute is defined as being equal to a flow of 24 cu. ft. per min. or 0·4 cu. ft. per sec. All operators are strongly advised to use only the cubic foot per second as the basis of their calculations. They are thus dealing with an accepted form of measurement readily convertible to other standards if desired and certainly not open to any misunderstanding.

The Venturi Meter.—This is a very simple and accurate arrangement for measuring the discharge through a pipe. It is standard for permanent installation on pipes from 3 or 4 inches diameter and over. The meter consists simply of a gradual reduction in the size of the pipe to a throat half to one-third the normal diameter of the pipe and a gradual expansion back to normal. It represents a sort of hour-glass shaped section with smoothly rounded internal walls into which are fitted two gauges (see Fig. 17). The planes of the internal openings of the gauge pipes are at right angles to the direction of the current.

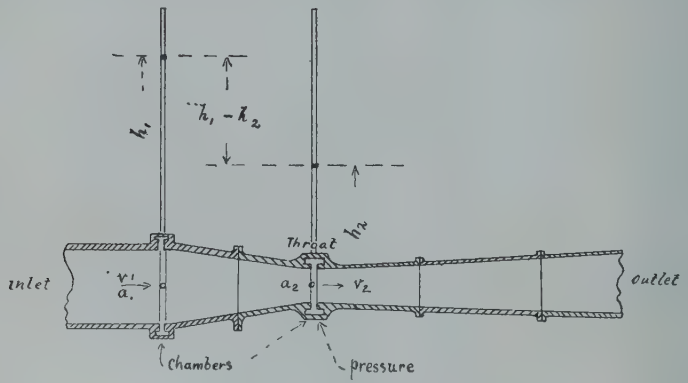

FIG. 17.—THE VENTURI METER.

The formula for discharge is based on the proposition that $h_1 + v_1^2 \div 2g = h_2 + v_2^2 \div 2g$ and that $q = a_1 v_1 = a_2 v_2$. This gives the formula :—

$$q = C \frac{a_1 a_2}{\sqrt{a_1^2 - a_2^2}} \cdot \sqrt{2g (h_1 - h_2)}.$$

Where a_1 and a_2 are the areas of pipe and throat respectively and h_1 and h_2 pressure heads at the pipe and throat, g — acceleration due to gravity and C = a co-efficient for friction varying from 0·94 to 1·0 (usually taken at 0·99). A manometer is generally used to measure h_1 and h_2, but pressure gauges may be used and their readings converted to feet of head, a correction being made for the height of the gauge above the centre line of the meter.

PIPELINES

Hydraulic Gradient.—If a pipe of uniform cross-section be connected with a dam, and water allowed to discharge from its open end, it has been observed that the pressure on the pipe

at any point is equal to the vertical distance between the centre of the pipe at that point and an imaginary line which is called the hydraulic gradient. Such a line is one drawn from a point slightly below the surface of the water in the dam to the centre of the outlet of the pipe. A pipe following such a line would carry exactly the same water as when laid horizontally from the bottom of the dam, but there would be practically no internal pressure tending to burst the pipe at any point. That a pipe may have a maximum discharge it is essential that no point in the line rises above the hydraulic gradient. Should it do so, that portion above the gradient acts as a siphon and the delivery is greatly reduced. Moreover the pressure in such a section is less than atmospheric.

In Fig. 18 the pipe rises above the hydraulic gradient A B and in this case a new gradient A F must be established and the flow calculated for this particular head. The section F G C merely acts as a siphon to carry off the water delivered at A F and if an opening were made at G not only would it not discharge water but air would on the contrary enter the pipe. The figure referred to admirably depicts an actual case of bad pipeline design with which one of the authors is acquainted. In the high ground a dam A was built to supply water under pressure for hydraulic operations at B. The pipeline followed the profile of the country as shown. The delivery at B was wholly inadequate. A careful survey of the line gave the profile as drawn and showed that the pipeline rose above the hydraulic gradient at F, G, C and E. At G where a nearly level section of the line dipped downward into a ravine, the line was cut and a surge tower G of ferro-concrete was erected. The tower was greater in height than the dam (12 ft.) and had a permanent stand-pipe opening to the air as shown and a sand trap in the base, on the opposite side of which was the offtake or continuation of the pipeline. The result of this was as follows :—A new hydraulic gradient H B was established above which no point in the pipeline rose and as a consequence it was possible almost to double the useful effect of the pipeline.

Design of Pipelines.—In no part of hydraulic mining is there observed a greater inefficiency than in the design and erection of pipelines. The authors can recall many examples which display only the rudiments of engineering design and the principles of hydraulics. Too often the pipeline is added as the last step in bringing water to the working face and with an utter disregard for the contour it must follow. Sharp bends and angles should be avoided and rapid reduction in the size of the pipes. Above all, the line should be so laid that no point rises above the hydraulic gradient.

The first consideration in design is to have a pipe of sufficient size to carry an ample margin of water over and above immediate requirements—in other words, permit of expansion of operations. The velocity of flow in the pipe should not exceed 5 ft. per sec.—less is better. A mean average flow of 3·5 ft. per sec. is an accepted figure. Above this the friction losses increase rapidly. A very slow flow, on the other hand, is bad, as it permits the growth of slime, fresh water algæ and rust, which destroy efficiency by increasing frictional resistance. As a rule it is best to calculate the carrying capacity of any pipe by formulæ designed for pipes with a rough internal surface ; for all pipes must by usage become more or less rough and offer greater resistance to the flow of water.

Due regard should be given to the strength of the material used in construction. The larger diameter pipes at the intake can be of lighter material than those at the discharge end of the line, where the pressure is greater. The bursting pressure should be calculated by any of the recognized engineering formulæ and applied with a reasonable factor of safety, for it should be remembered that in mining work—particularly in the working face where the pipes are subject to frequent moves—hard usage has to be borne by all material. Falls of earth or timber that will ruin a light pipe would be resisted if the pipe were of stouter material. Large pipes, therefore, should not be made of too light material. They may be sound enough from an engineering point of view, but too light to withstand damage in transport, particularly in remote countries without proper facilities and erection on rough ground. If the pipeline has to be transported long distances before erection, consideration should be given to the adoption of such sizes as will facilitate nesting for shipment without affecting efficiency. Seldom is it justified in mining operations to procure rolled pipe sections and rivet them on the job.

Details of pipe design are often matters of individual taste. Some operators prefer flanged pipes—straight or spiral riveted—whilst others adhere as much as possible to the slip-joint types. For hydrostatic heads not exceeding 300 feet, the authors prefer the slip-joint pipe with lugs. With greater pressures than this they consider flanged pipe more suitable. The use of spiral riveted pipe is in most cases uncalled for. There is an excessive cost with little compensating benefits. Once damaged their repair is most difficult without machine shop facilities. The slip-joint pipes when being coupled readily conform to the surface of the ground and appreciable curves can be made merely by the flexibility of the joint.

Long pipelines which are laid on the surface (which is commonly the case for mining purposes) should be well provided with expansion joints to compensate movements due to changes

of temperature. All pipelines should be supplied with air valves which should be roomy and of simple design. They should be

FIG. 19.—PIPELINE SHOWING ANCHORAGES.

fitted at all high points on the line. They perform the dual service of allowing air to escape when the line is being filled and admit air to the line to avoid collapse in the event of a pipe bursting.

Pipes laid on inclined positions need careful anchorage to avoid movement. The methods vary with the size and permanence of the installation and may either be clamps surrounding the pipes and fastened to steel pins in the ground or may be masonry or concrete supports in which the pipe is embedded (Fig. 19).

Where inverted siphons are used, provision should be made for their drainage.

The head of the pipeline should junction with the penstock box by means of a liberal bell-mouth pipe, entry to which is controlled by a gate valve (Fig. 20). An air valve should be placed at the head of the line to admit air as the gate valve is closed. A stand-pipe always open to the air is favoured by some operators

Fig. 20.—A Bell-Mouth Pipe leading from the Penstock shown in Figs. 23 and 24.

Filling Pipelines.—In filling a pipeline for the first time care should be taken not to admit the water at too great a rate otherwise shock from included air or contraction may cause severe damage. New pipelines, when first put in use, often show numerous small leakages ; these can often be stopped by feeding in small quantities of sawdust at the intake.

Expansion Joints.—When a pipeline is submitted to extreme changes of temperature provisions should be made for the insertion of expansion joints in suitable positions. Care should be taken that the female end of the joint is well anchored.

The Penstock or Pressure Box.—When a pipeline is brought direct from a dam a penstock is not required and it is only necessary to make provision to eliminate the possible entry

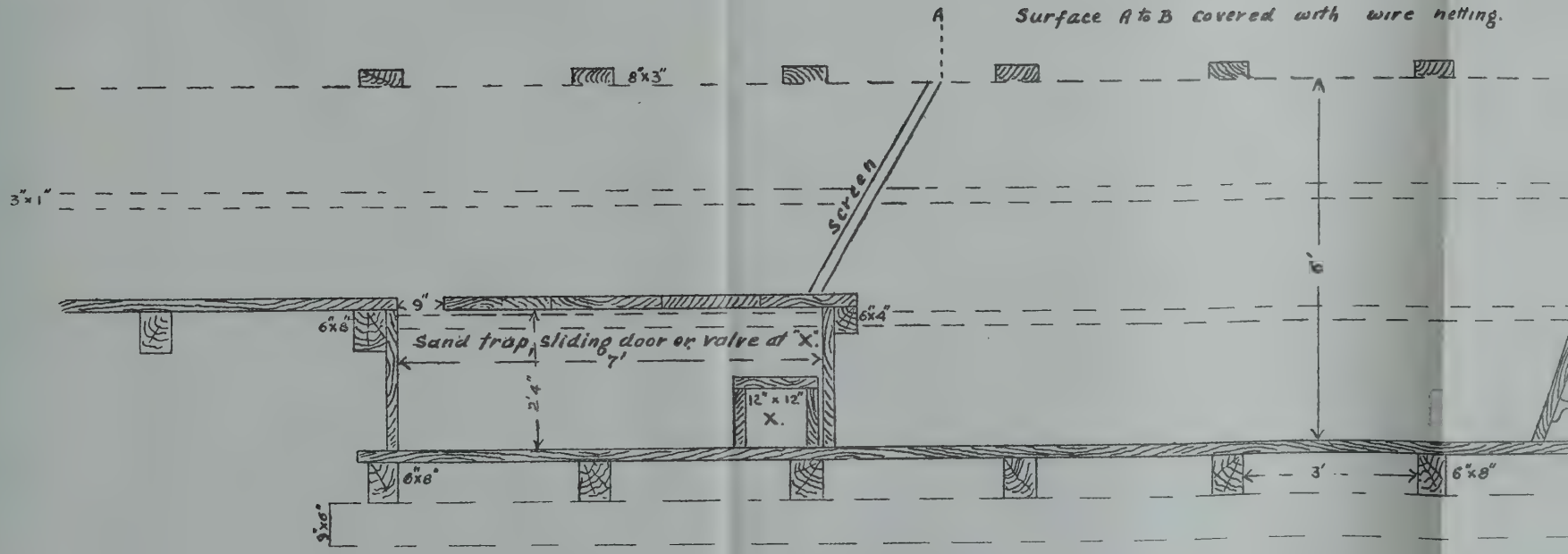

FIG. 21.—CROSS SECTION OF A PENSTOCK OR PRESSURE BOX.

of timber and sand. Good screening arrangements and a sand trap will overcome these difficulties. Sometimes, however, a dam is merely used as a means of diverting part of the water of a stream to a ditch line in which case the water is drawn off at the surface of

Fig. 22.—Illustrating a Penstock or Pressure Box.

the dam. Here, again, the forebay or entrance to the ditch should be designed to exclude the entry of timber and sand. Screens will meet the former contingency whilst accumulations of sand are removed from the dam by a sand or sludge door placed at the

Fig. 23. –Ferro-Concrete Penstock in Course of Construction.

lowest point in the retaining wall. In the case of limited supplies of water the sludge door is periodically opened but where ample water avails, it is often left open continuously.

When water is conveyed in a ditch to a pipeline the ditch terminates in a penstock or pressure box. This generally is of

FIG. 24.—ANOTHER VIEW OF THE PENSTOCK IN FIG. 23.

FIG. 25.—HYDRAULIC MONITORS.

rectangular form and may be constructed in wood, iron, or concrete according to the magnitude of the work, the nature of the ground,

and the life of the mine. Wood and iron structures conform with most of the conditions met with in hydraulic mining, but where power is developed concrete work is more in favour. In all cases ample provision should be made to prevent timber, floating reeds, leaves, sand, and even fish, from entering the pipeline. Arrangements should also be made for directing excess water to a by-pass for delivery well away from the penstock. Figs. 21–4 show different types of penstock.

The penstock should have ample depth for unless the inlet to the pipe column has good submergence air will enter with the water,

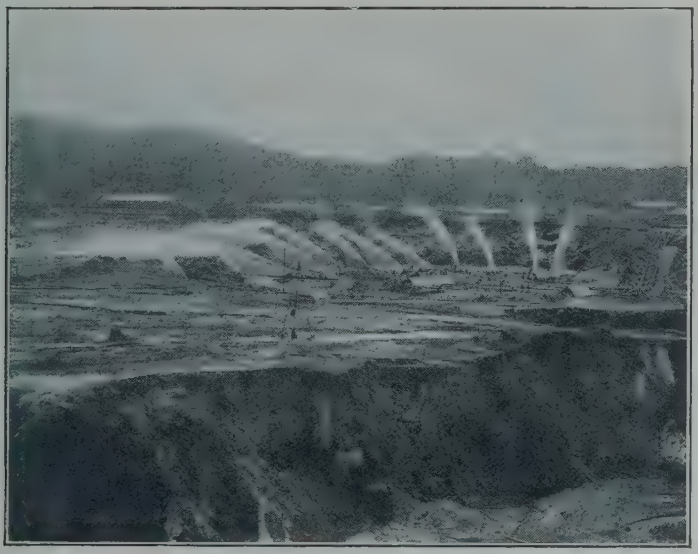

Fig. 26.—Hydraulic Monitors.

and will later cause trouble at the nozzles by causing the jet to break or spread.

A sand-trap is placed across the bottom of the penstock in such a position that it can be periodically flushed out. If much sand finds its way into the pipe column it causes serious wear in the pipes and does much damage to the seats of the gate valves as well as to the nozzle tips.

Nozzles and Monitors.—The monitor is the final but by no means least important detail in the design of an hydraulicking plant. Its size will depend on the volume of water and the effective head. It may merely be required on the one hand as a

cutting and disintegrating agent when additional water as a carrying medium is available from other sources for moving the broken material forward along the sluices; on the other hand it may have to supply sufficient water to both cut and transport the material. Furthermore, having regard to pressure available and depth of ground being worked, it may in one case be desirable to so design the nozzle that it will throw its jet of water as far as possible in an unbroken stream, particularly in deep ground where danger exists in operating too close to the working face, or in other cases where ground is shallow and easily dis-

Fig. 27.—Hydraulic Monitors. Shows also sluice boxes brought right up to working face.

integrated a nozzle throwing a slightly broken jet will most effectively break the ground without merely "boring" it. For high pressure, nozzles are best designed with a good long snout, the shorter or bull-nose type being reserved for lower pressures. (Figs. 25–27.)

The barrel of the monitor should have good long water vanes along its interior to put the water in train before it reaches the nozzle. These guide vanes remove the tendency of the water to spin and prevent the jet breaking after leaving the nozzle tip. Good submergence should be given to the intake pipes at the

TABLE 9

Table showing the Quantity of Water discharged, in Sluice-heads, or Cubic Feet per Second, from Different Diameters under Different Heads of Water.

Head on Nozzle, in Feet.	Diameter of Nozzle in Inches.													
	1	1¼	1½	1¾	2	2¼	2½	2¾	3	3½	4	4½	5	5½	6
	Sluice-heads discharged per minute.														
10	0·129	0·202	0·290	0·394	0·517	0·650	0·808	1·000	1·202	1·636	2·137	2·702	3·339	4·041	4·808
20	0·183	0·285	0·410	0·560	0·730	0·925	1·144	1·385	1·700	2·314	3·022	3·822	4·722	5·714	6·800
30	0·224	0·349	0·504	0·688	0·896	1·136	1·400	1·696	2·082	2·834	3·699	4·681	5·784	6·998	8·328
40	0·259	0·405	0·581	0·792	1·034	1·309	1·626	1·958	2·404	3·272	4·274	5·403	6·678	8·080	9·617
50	0·290	0·453	0·650	0·906	1·157	1·406	1·813	2·190	2·688	3·659	4·779	6·043	7·467	9·035	10·753
60	0·317	0·496	0·712	0·970	1·269	1·605	1·984	2·399	2·944	4·007	5·234	6·620	8·178	9·896	11·719
70	0·344	0·536	0·770	1·048	1·370	1·738	2·141	2·591	3·180	4·329	5·654	7·150	8·834	10·609	12·721
80	0·365	0·573	0·824	1·122	1·464	1·853	2·288	2·761	3·389	4·613	6·025	7·619	9·413	11·390	13·553
90	0·387	0·605	0·874	1·189	1·581	1·968	2·426	2·938	3·606	4·909	6·411	8·106	10·017	12·121	14·425
100	0·408	0·640	0·922	1·256	1·640	2·079	2·560	3·098	3·801	5·175	6·759	8·547	10·560	12·777	15·207
110	0·429	0·672	0·965	1·312	1·717	2·173	2·693	3·224	3·987	5·386	7·084	8·895	10·993	13·299	15·834
120	0·448	0·701	1·008	1·392	1·794	2·272	2·805	3·393	4·164	5·668	7·403	9·362	11·567	13·996	16·657
130	0·467	0·728	1·050	1·429	1·866	2·363	2·917	3·523	4·334	5·900	7·705	9·744	12·040	14·568	17·336
140	0·482	0·757	1·090	1·485	1·938	2·453	3·029	3·665	4·498	6·123	7·997	10·112	12·494	15·120	17·992
150	0·501	0·784	1·128	1·536	2·005	2·536	3·136	3·794	4·655	6·336	8·276	10·468	12·931	15·640	18·622
175	0·540	0·845	1·219	1·664	2·168	2·737	3·386	4·098	5·029	6·844	8·940	11·306	13·969	16·902	20·144
200	0·580	0·904	1·306	1·773	2·317	2·931	3·621	4·381	5·376	7·317	9·577	12·084	14·933	18·070	21·505
250	0·648	1·010	1·453	1·984	2·589	3·280	4·048	4·897	6·011	8·181	10·686	13·514	16·696	20·203	24·043
300	0·710	1·106	1·600	2·173	2·837	3·590	4·434	5·365	6·585	8·962	11·705	14·802	18·290	22·130	26·337
350	0·766	1·197	1·724	2·346	3·065	3·880	4·789	5·795	6·896	9·387	12·268	15·518	19·157	23·180	27·586
400	0·819	1·280	1·843	2·509	3·277	4·147	5·123	6·195	7·378	10·034	13·107	16·589	20·480	24·781	29·491
450	0·869	1·357	1·955	2·661	3·476	4·399	5·430	6·571	7·820	10·512	13·901	17·595	21·727	26·282	31·280
500	0·916	1·431	2·061	2·805	3·664	4·594	5·724	6·927	8·243	11·455	14·654	18·547	22·897	27·706	32·792
550	0·960	1·501	2·161	2·942	3·843	4·863	6·004	7·265	8·645	11·768	15·368	19·452	24·015	29·058	34·582
600	1·003	1·568	2·257	3·073	4·013	5·079	6·300	7·588	9·030	12·291	16·052	20·318	25·083	30·350	36·150
650	1·044	1·632	2·350	3·198	4·177	5·286	6·527	7·896	9·398	12·793	16·708	21·147	26·107	31·590	37·594
700	1·084	1·693	2·438	3·318	4·335	5·406	6·773	8·195	9·753	13·275	17·338	21·946	27·092	32·782	39·012

[page 43

penstock to prevent as far as possible the inclusion of any air with the entering water. Such included air will on being discharged at the nozzle tend to break the stream and destroy its concentrated hitting or cutting power. Where high pressures are available the nozzles should be fitted with deflectors as an aid in directing the jet. Low-pressure nozzles do not require deflectors.

The quantity of water discharged by the nozzle may be calculated by the following formula :—

Where g = acceleration due to gravity ;
h = the head in feet ;
a = area of nozzle in feet ; then the discharge in cubic feet per second equals

$$0{\cdot}98 \sqrt{2gh} \times a.$$

DITCHES, WATER RACES, OR LEATS.

Survey and Design.—The conveyance of water in open channels is covered by the various terms of ditches, leats or water races, and refers more particularly to mining work. For the larger work of irrigation and hydro-electric schemes the term canal is more frequently used. In mining operations such works are as a rule designed for ground-sluicing, hydraulicking, jet elevating, or direct power production by means of Peltons or turbines.

Whatever the object in view the engineer-in-charge should make most careful observations. The natural contour of the country may make a certain intake for the ditch imperative and the ditch line must then be constructed so as to secure the required amount of water at the maximum available head unless a low head is of service, but it is good practice always to retain within reason all the available head. So also is it good practice to design the ditch line to deliver more than immediate requirements and thus have a margin in hand for possible expansion of operations. Allowance must also be made for evaporation and seepage losses. Physical conditions or even legislation may preclude taking such precautions, but where possible advantage should be taken of them.

On the other hand the contour of the country may make possible the securing of a suitable intake for the ditch a matter of particular ease, while requirements demand a minimum head. In the one case then " flying " levels should be taken from the intake, and in the other from the desired penstock position. In either case the gradient will be governed by the nature of the ground through which the ditch must be constructed. After the " flying " levels have been recorded a rough detailed levelling

should be done to ascertain the approximate length and route of the ditch so as to afford information concerning the amount of cutting and filling, whether flumes are necessary, or even tunnels, siphons or constructional work, to get around rocky points. The course should be staked at every chain or 100 feet with a short peg driven well into the ground. A longer and pilot peg at every fifth station is advantageous. The levelling will, of course, have been done on a gradient suitable to the ground and as dictated by the engineer's experience. Having done this the engineer has fortified himself with a wealth of information. He should now re-traverse the course of the ditch and note those places where the bends and turns in the preliminary line are too sharp and should lay out his work, maintaining as much as possible long straight lines and wide curves. For instance levelling at 100 ft. distances small variations along an apparently level stretch of country will give him a line somewhat after that shown in A to J in Fig. 28.

FIG. 28.—LAY-OUT OF A DITCH.

There is no reason whatever to follow such a course. He should re-align somewhat after the manner shown by the dotted course. He thus eliminates many bends and turns in the ditch which when completed will afford the water a smooth even flow with a minimum of friction. The same adjustments apply to curved sections in the ditch line. The authors emphasize this point because it is within their knowledge that many such constructional errors have been made after the engineer has gone to much previous trouble in calculating the correct gradient and capacity for his ditch. Careless construction has nullified the attention and care given to calculation before the inception of the work, and he finds his ditch only delivering a portion of the quantity he calculated to receive at the outlet. In the case of ditches in districts liable to heavy rains, ample provision should be made for spillways along their course so as to pass excess water away and avoid damage by overflow in undesirable places.

Gradient for Ditches.—The gradient will be regulated by the nature of the ground, care being taken that the velocity of flow is insufficient to erode the bottom or sides of the ditch. Some ground will stand a velocity of flow equal to 5 or 6 ft. per sec. whilst in softer soils less than half this velocity will cause scouring. A good average gradient is 8 ft. per mile, but the authors have seen as low as 2·6 ft. per mile used in flat country

and as much as 32 ft. in more hilly regions. Good practice favours a velocity of flow of 3 ft. per sec. The gradient will then depend on the dimensions of the ditch and the quantity of water it will be required to deliver. All ditches when newly constructed have a greater seepage loss. In an old ditch sediment and the growth of fresh water algæ will tend to reduce this but never eliminate it. If the velocity of flow is too slow, the growth of aquatic reeds may hinder flow.

Carrying Capacity of Ditches.—There are numerous formulæ for calculating the carrying capacity of open channels.

Eytelwein's formula is :—The mean velocity of the water in feet per second equals the square root of (twice the fall in feet per mile multiplied by the hydraulic mean depth); but this gives rather high results for ditches.

The Neville formula viz. $V = 140\ \sqrt{RS} - 11\ \sqrt[3]{RS}$ gives results much the same as Eytelwein's.

The closeness with which these formulæ approach the actual discharge of a ditch depends entirely upon the nature of the ground and the correct construction of the ditch. The divergent nature of the material in earth cuttings makes it a practical impossibility to give a formula for mean velocity to correspond with actual discharges and the best method is to make allowance for ditches which follow a sinuous course. The authors consider that the Gordon formulæ more closely conforms to the conditions met with in mining and in which he gives $5a\ \sqrt{2grs}$ = carrying capacity for ditches having a sinuous course and $6a\ \sqrt{2grs}$ = carrying capacity where the earthwork and cuttings have a tolerably straight course. The notation of the formula is given below :—

g = acceleration due to gravity ;
r = hydraulic mean depth ;
a = the sectional area ;
s = the sine of the angle of inclination.

FLUMES.

Flumes for mining purposes are at most times a necessary evil, and unless the life of the mine warrants elaborate construction they are a continual source of trouble. In most cases construction in wood only is justified and in certain climates this is liable to rapid decay, destruction by white ants, and damage by storm or fire. In dry climates where the flow of water decreases considerably in the summer season they need constant repair to overcome the effects of warping and cracking caused by the heat of the sun. Where it is reasonably possible to obviate their

use by extending the ditch-line into the ravine on one side and conducting it out again on the other, this is favoured by the authors. In the ravine, stream, or depression, as the case may be, some simple form of overflow dam makes a serviceable by-pass for the water and at the same time is a useful spillway for excess water when the ditch is at full carrying capacity.

Fig. 29.—Flume in Course of Construction.

Flumes are usually constructed of timber rough from the saw-bench and not dressed in any way. They are commonly rectangular in section. Where metal is used, their construction follows the semi-circular and trapezoidal forms and is advised where conditions warrant, such as the life of the property and the value of the water in cases where water is purchased. With such structures losses of all kinds are at a minimum.

In flumes the velocity of flow of the water is greater than in channels constructed in earthwork and in their design it is usual to give them a slightly less gradient otherwise "crowding" of

FIG. 30.—THE SAME FLUME IN COURSE OF CONSTRUCTION.

FIG. 31.—THE SAME FLUME COMPLETED.

the water takes place at the delivery end of the flume, destroys the "train" of flow, and may cause scouring. (See Figs. 29–32.)

Carrying Capacity of Flumes.—For wooden flumes constructed with sawn timber (undressed) and having battens over the joints

inside the flume the carrying capacity can be calculated with reasonable accuracy by the following formula :—

$$8\sqrt{2grs} \times a = 64{\cdot}16 \times a \times \sqrt{rs}$$ = cu ft./sec

where $g \times$ acceleration of gravity ;
r = the hydraulic mean depth in feet ;
s = the sine of the angle of inclination ;
a = the area of flume, in square feet.

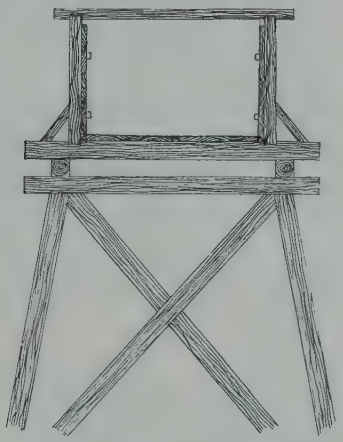

Fig. 32.—An Elevated Flume.

With a flume 4 ft. wide and two ft. deep on a gradient of 24 ft. to the mile, the quantity of water that the flume would deliver would be as follows :—

$8\ (2grs)^{\frac{1}{2}}$

$$2g = 64{\cdot}4$$

$$r = \frac{\text{area}}{\text{perimeter}} = \frac{8}{8} = 1$$

$$a = 8 \text{ ft.}$$

$$s = \frac{24}{5280} \cdot \qquad = 0{\cdot}004545$$

Therefore $64{\cdot}4 \times 1 \times 0{\cdot}004545 = 0{\cdot}29273$

$$\sqrt{0{\cdot}29273} = 0{\cdot}54101$$

$0{\cdot}54101 \times 8 \times 8 = 34{\cdot}627$ cubic feet per second or $34{\cdot}627$ sluice-heads per minute.

This formula is designed for flumes of reasonable length and is not applicable for very short flumes such as are used in ditch-line connections. For such flumes it is quite safe to use the ordinary ditch formulæ.

METHODS OF WORKING GROUND.

The earlier methods of working cannot of course be classed as of economic importance, but they were steps towards modern

FIG. 33.—NATIVES PANNING OR CALABASHING TIN-BEARING WASH.

FIG. 34.—ANOTHER VIEW OF NATIVES PANNING.

practice. First were the Batea and gold pan (Figs. 33 and 34), then came various types of "Rockers" (Fig. 35, single and double

deck) and the "Long Tom" (Figs. 36 and 37). The latter is virtually the forerunner of the modern sluice box. Both the Rocker and Long Tom are still useful accessories in testing deposits in very isolated places. They can be readily made on the spot and are most useful in treating small bulk samples such as the whole of the contents of a sample pit.

Ground Sluicing.—This is the simplest form of alluvial mining and is generally applied in cases where deposits occupy the hillside well above the existing stream level or where the bedrock of the deposit adjacent to the stream is only slightly above that level.

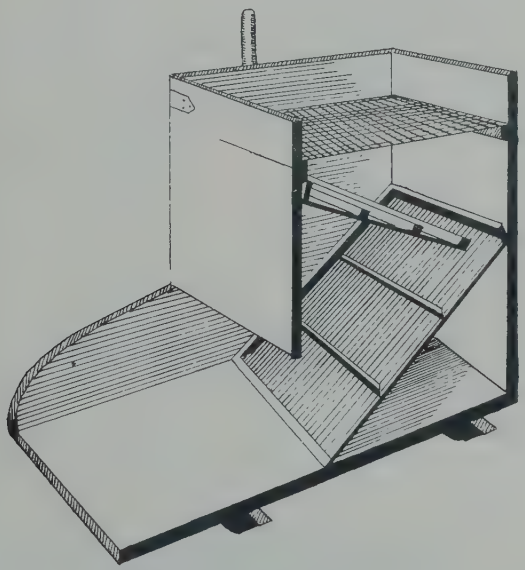

FIG. 35.—A TWO TRAY ROCKER WITH SIDE REMOVED TO SHOW THE METHOD OF CONSTRUCTION.

The water is usually brought on to the work by means of a ditch from which it is drawn and used by simple gravitational means. The preliminary operation (after completion of the ditch) consists of opening out the main tail-race or sluice as shown in Fig. 38. A trench (AC) is first cut and then water from a feeder race AB is turned into it. Workmen with picks or mattocks then commence cutting it down to grade level and width. The flow of water considerably aids in the transport of the material into the river as well as softening the material for disintegration.

This sluice is laid out in such a way as to take advantage of the general slope of the ground as indicated by the arrows, and is cut down to bedrock or if possible a few feet into the bedrock (see also Figs. 39–42).

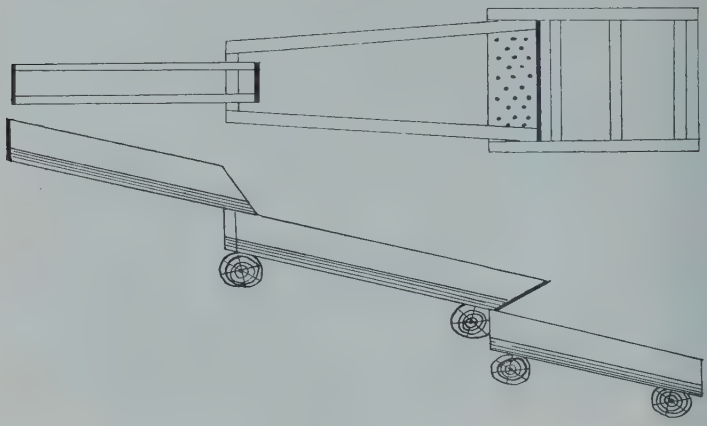

Fig. 36.—The Long Tom.

The next step is to open out the paddocks or working faces. Details of procedure in this regard will be dictated by the nature and depth of the ground. In deep ground one might proceed in benches, or where the top alluvium is valueless and consists for the most part of soil, sand, and fine grit it may be advantageous

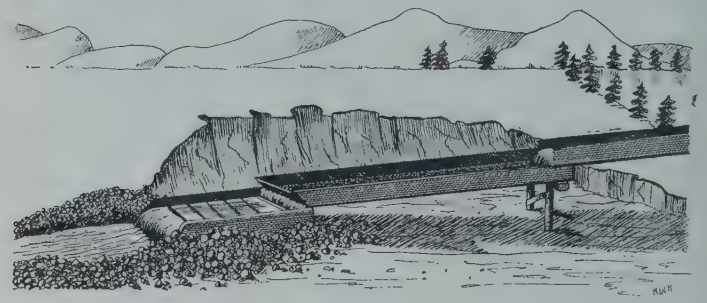

Fig. 37.—Another idea of the Long Tom.

to strip this off first and pass it away through the sluice as quickly as possible. In either case the initial procedure will be as follows: Water is brought to the main sluice somewhat in the manner indicated in Fig. 38 at D and the ground excavated by pick or mattock aided by water brought over the edge of the face as

shown by the sub-divided races. Excavation proceeds in the direction of XD again taking advantage of the general slope of the ground to assist the water in transporting the broken material to the main sluice. Over a wide face this is not easy, since the water tends to spread and deposit material along the floor of the paddock unless kept in bounds by stone or temporary banks of earth. A very effective means is provided by the use

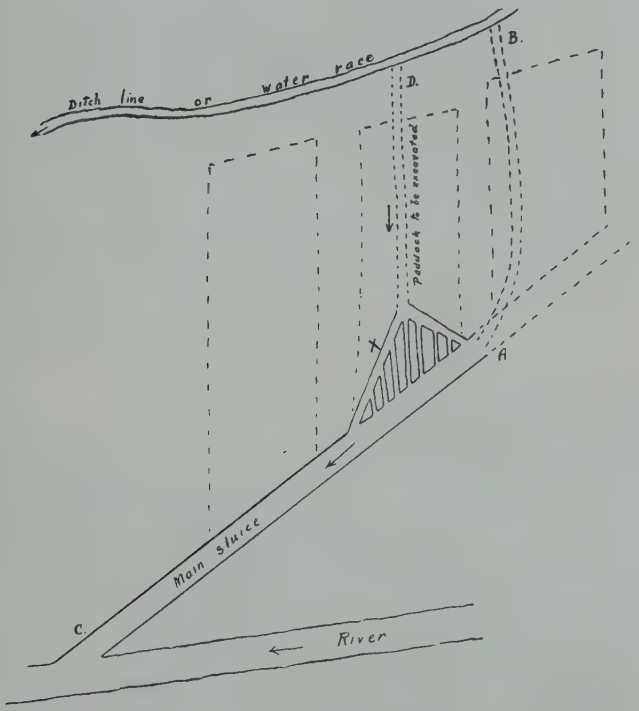

FIG. 38.—LAY-OUT FOR GROUND SLUICING WORK.

of face boards or planks stood on edge in such positions as to confine the material and form a temporary sluice as it were. A useful design of face board is shewn in Figs. 43 and 44. The two iron wings are driven into the soft bedrock to hold it in position.

Where topographic conditions are favourable, ground sluicing methods may be improved or extended by the use of water under pressure, i.e. in the use of monitors for directing water against the "face" of material to undermine, disintegrate, and drive it

into the sluices (Figs. 45 and 46). With still more favourable conditions greater pressures will be available and the simple

FIG. 39.—GROUND SLUICING IN A FLAT POSITION. Overburden has been removed and natives pick up the uncovered wash and convey it to a sluice.

FIG. 40.—OPERATIONS SIMILAR TO THOSE IN FIG. 39.

ground sluicing operations thus merge into one of hydraulicking only or of hydraulicking and elevating. It will thus be seen that

FIG. 41. A GROUND SLUICE IN OPERATION (Sluice cut in the ground).

FIG. 42.—A SIMILAR SLUICE IN OPERATION.

topography plays an important part in the determination of a method of operation. In any sluicing problem the main object (apart from effective recoveries) should be the attainment of the fullest use of both the natural slope of the ground and the quantity of water available. Effectively applied these two factors give one hundred per cent service, but their successful use depends on the personal element. Continual attention is necessary to secure the best results.

Ground Sluicing and Hydraulicking.—This method of working, as the title signifies, simply consists of the use of water under

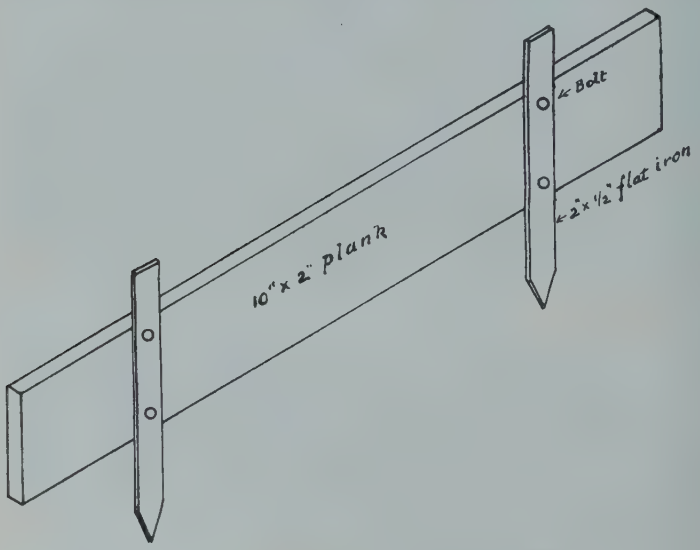

Fig. 43.—A Face-board used in Ground Sluicing to Confine the Flow to a Definite Course.

pressure to break and transport the material. The use of water under pressure (in a nozzle) (Fig. 47) will imply a reduction in the labour force as well as in the equipment of the paddock. Face-boards will not be necessary since any accumulation of material can be readily driven into the sluices by turning the monitor round and "blowing in" the disintegrated alluvium. In hydraulicking deep ground high pressures are necessary since it is unwise to place the monitor too close to the operating face. A fall of ground in a deep face has been known to rush forward on a too-closely situated monitor and bury both it and the operator (Figs. 45 and 46.)

Hydraulicking and Elevating.—This combination represents conditions apart from ground sluicing. It implies the existence of a deposit below the natural gradient of the streams, and which therefore has to be lifted to sufficient height to facilitate its economic disposal. The success of this type of work is dependent on a cheap and plentiful supply of water at an effective pressure. The mechanical efficiency of such plants is quite low, but this is compensated for by their low capital cost, simplicity, and cheapness in operation (Fig. 48).

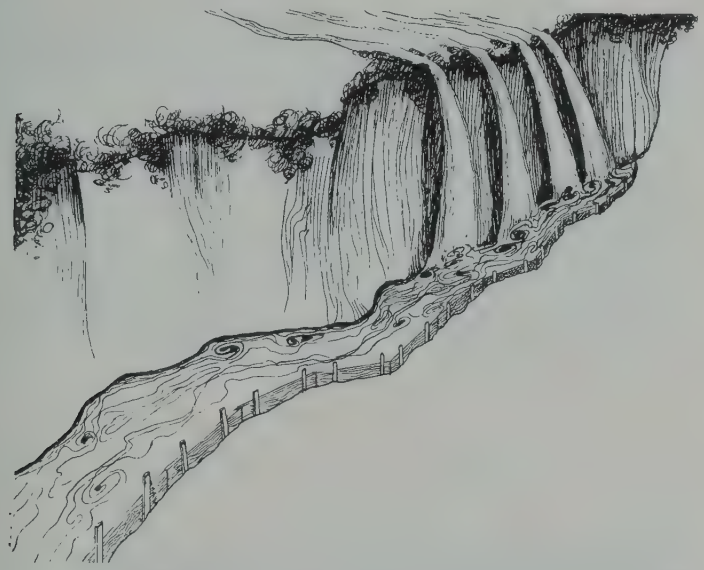

FIG. 44.—FACE-BOARDS KEEPING THE FLOW OF WATER AGAINST THE WORKING FACE.

Water Lifters.—The same principles as applied in gravel elevators are employed in water lifters. They will operate either submerged or otherwise; when submerged, however, their efficiency is much lower unless some provision is made for the admission of air. This is best done at a point above the smallest point of contraction in the throat where in ordinary circumstances there is a tendency for a partial vacuum to form. The admission of this air greatly assists the elevator and virtually converts it to a hydro-pneumatic lift. With water-lifters care should be taken to have the control valve at some point above the probable level of unexpected floods in the sluicing paddock. The air inlet if any should also be raised above probable flood level (Fig. 49).

THE SLUICE OR SLUICE BOX.

Where sufficient gradient is available for the disposal of tailings the sluice is often constructed in the ground. A

FIG. 45.—WATER UNDER PRESSURE BEING USED TO BREAK GROUND AND DRIVE IT INTO THE SLUICE.

FIG. 46.—WATER UNDER PRESSURE IN USE AS IN FIG. 45.

rectangular cut is made in the ground and through this is passed all the alluvium from the working faces. More often they are

built up on trestles either in iron or wood and as they are periodically moved from place to place to keep up with the progress of the working faces they are usually constructed in sections so that they may be readily dismantled and re-erected on a new site with a minimum of delay and damage (Figs. 50–2).

The size of the sluice depends on the quantity of water available for operating and the gradient. Many favour the narrow deep sluice as being more easily kept clear. This is quite erroneous since a wide sluice will carry more material with less grade and save considerably more gold or tin, as the case may be. The whole object should be to pass the material through the " box " in as thin a layer or film as possible so that the gold or tin has every opportunity to settle as rapidly as possible (Fig. 53).

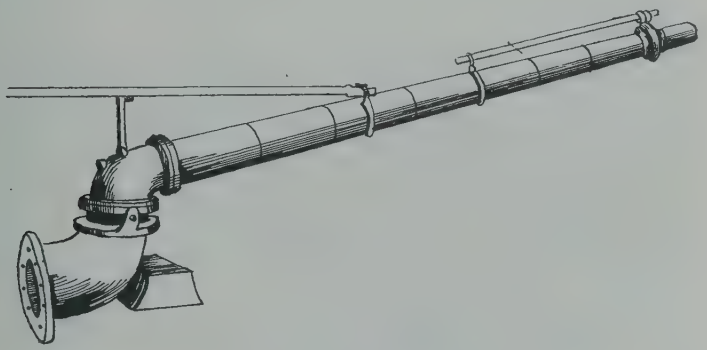

Fig. 47.—The Hydraulic Nozzle.

A sluice should be straight, but if curves must be introduced they should be of wide radius and the outer side of the sluice should be slightly raised. When the material is cemented, the sluice should have several drops to aid in breaking up the lumps. Clay, however, is the worst material to deal with. It forms balls and as it rolls through the sluice will pick up gold or tin and carry it away to the tailings dump. Auxiliary sluices are often provided into which the clay is forked as soon as discovered and periodically flushed direct to the dump.

In a sluice set on a gradient of 1 in 25, one cubic foot of water per second will wash away about 6 to 8 cubic yards of gravel per hour according to the nature of the material. Fine material is the most difficult. A reasonable amount of stone helps considerably. It keeps the mass lively and prevents packing.

Construction and Design of Sluice Boxes.—Sluice boxes are more commonly constructed in wood or steel, or a combination

of both. In the latter case an angle iron frame carries steel sides and a wooden bottom. Both this and the "all steel" type are usually made in sections 5 ft. long to facilitate handling and erection. The purely wooden type is only in favour where plenty of timber is available close to the mine and in circumstances where they can be erected and kept running for a long time in the one position before dismantling and re-erection become necessary with the extension of mining operations. They have the further drawback of warping and cracking in dry climates if left empty, and in tropical countries are damaged by white ants.

FIG. 48.—PANORAMA OF A HYDRAULIC AND
The elevator is at the head of the

The sectional steel box is most favoured. In the smaller sizes (up to 3 ft. wide and in which the sides do not exceed 15 inches in depth) they are usually pressed in one piece. U-shaped angle irons riveted at each end facilitate their coupling together. In the larger sizes the bottoms and sides are made separately and coupled on longitudinal angle irons as well as at the ends. Sometimes the top edge of the side plates is also reinforced with light angle iron. The sides in the larger sizes rarely exceed a depth of 2 ft.

Sluice boxes are rarely placed flat on the ground and are generally erected on trestling in order to give sufficient gradient for their operation and to afford a readier disposal of the tailings. The trestles may be constructed of squared or round timber frames and should be well braced. It is also an advantage to allow one side of the cap of the frame to have an overhang of about 3 ft. This allows of a walk being constructed alongside the boxes.

Dimension and Grade of Sluice Boxes.—The width of the sluice box has an important bearing on the success of the work accomplished. Its length is not nearly so important. In earlier

ELEVATING PLANT, ALSO SHOWING SLUICE BOXES.
sluice, but is not visible in the photograph.

tin sluicing operations the more common widths were 3 ft., 4 ft., and 6 ft. with a depth of 2 ft. Modern plants now have sluice boxes that vary in width from 8 ft. to 20 ft. and 2 ft. deep. The gradient will vary from 1 in 30 to 1 in 20. The figure more generally favoured by operators is 1 in 25. The length of the sluice may vary from 75 ft. to 300 ft. and in determining this one should be guided by the fineness of the cassiterite and the nature of the material, In a long sluice tough material has a better chance of disintegrating but very clayey material would only "ball"

and pass right through the box. It is good practice to step the box at intervals. This will help to break up material as well as assist in an even re-distribution of the material over the entire width of the sluice.

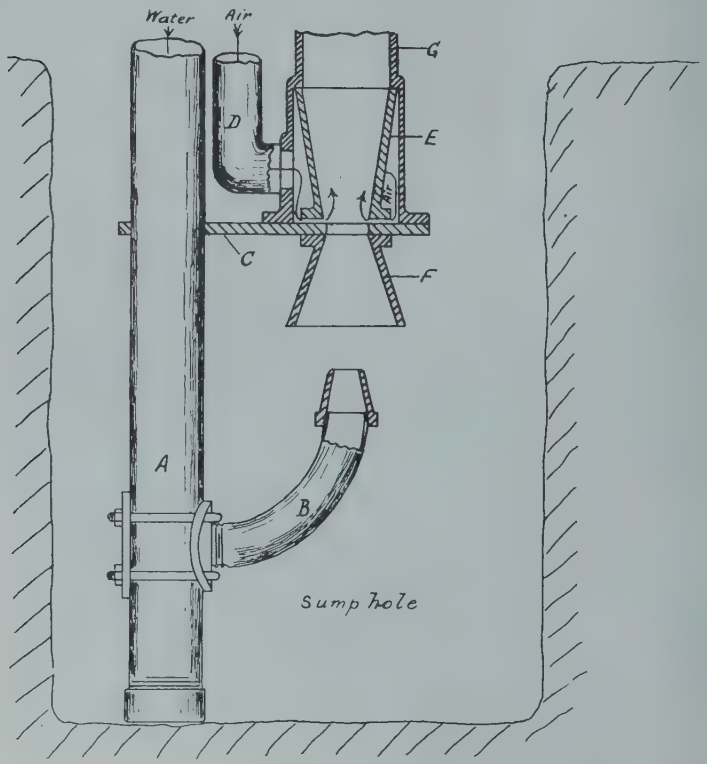

FIG. 49.—SKETCH ILLUSTRATING THE PRINCIPLE OF THE WATER LIFTER.

In determining the gradient of a sluice the following factor should be taken into consideration :—

(*a*) The nature of the material to be sluiced.
(*b*) The dimensions of the sluice.
(*c*) The velocity of the current required to sluice the material under treatment.
(*d*) The quantity of water available.

The following table is generally accepted as a most useful guide in determining the velocity of the current.

FIG. 50.—TRESTLING TO CARRY SLUICE BOXES IN AN ELEVATED POSITION TO ASSIST IN DISPOSAL OF TAILINGS.

FIG. 51.—TRESTLING FOR USE AS IN FIG. 50.

Water moving at the rate of :—

16 ft.	per	minute	begins to wear away fine silt.			
30	,,	,,	,,	just lifts fine sand.		
39	,,	,,	,,	carries fine sand.		
45	,,	,,	,,	moves fine sand as coarse as wheat.		
120	,,	,,	,,	moves pebbles	1 inch	diameter.
200	,,	,,	,,	,, ,,	$1\frac{1}{2}$ to 2	,,
320	,,	,,	,,	,, stones	3 to 4	,,
400	,,	,,	,,	,, ,,	6 to 8	,,
600	,,	,,	,,	,, ,,	12 to 18	,,

No hard and fast rule can be laid down in determining this velocity. The operator will be guided to a large extent by his own experience. It is here, however, that the beginner is faced with difficulty, and he would be well advised in adopting in the

FIG. 52.—TRESTLING IN USE AS IN FIGS. 50 AND 51.

first instance a 4% gradient for his sluice boxes and a minimum length of 120 ft. A careful sampling of the tailings will indicate the efficiency of his sluice.

The following formula has been given for determining the grade of the sluice :—

$$F = \frac{V^2 \times P}{2A} \times C$$

Where F = fall in ft. per mile.
V = velocity in ft. per second.
P = wetted perimeter of sluice.
A = area in sq. ft. occupied by water and dirt in the sluice.
C = a co-efficient which varies from 6 for light material to 8 for boulders and heavy clay.

The gradient in ft. per mile may be reduced to in. per 12 ft. box by multiplying by 0·027.

The beginner is not advised to rush to formulæ as an aid in determining the grade. At best they are only approximations. He will do much better by adopting what is standard practice, and at the same time carry out his own experiments. Every sluicing proposition is a problem in itself, and there are too many possible disturbing factors for any formula to be applied in a hard and fast manner.

The Paving of Sluices.—In the sluicing of auriferous alluvium it is usual to pave the sluice with stone or wooden blocks. Where there is plenty of timber wood is used. The blocks are set on end. In cleaning up they are readily removed and easily set in position

Fig. 53.—A Sluice in Operation.

again. Stone blocks, however, have a considerably longer life and by many are considered better gold savers. The blocks are usually 14 in. deep and in the case of wood will last barely two months (see Figs. 54 and 55). Alternative to blocks and sometimes in conjunction with them, cross riffles are commonly used. They consist of varied forms of angle iron and sometimes sections of old railway rails are used. They form a neat effective sluice. Sometimes they are underlain by fibre matting. Prospectors in Australia make quite an effective sluice by using the split trunks of tree fern, set cross-wise in the sluice with the fibre uppermost. In cleaning up—generally after a run of several months—the

fern trunk is burned and the gold recovered from the ashes. A gold sluice is generally narrower than a sluice for tin and much

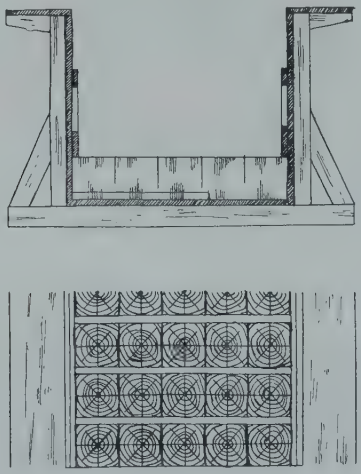

Fig. 54.—Sluice paved with Timber.

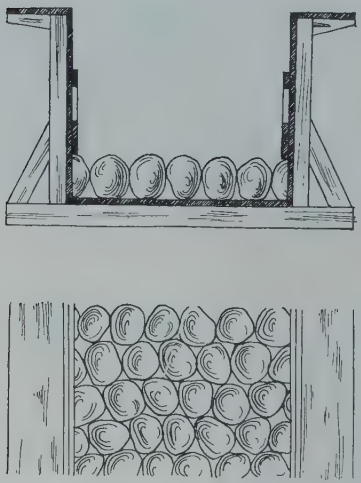

Fig. 55.—Sluice paved with Stone.

longer—sometimes several hundred feet long. The lower reaches of such sluices are often run for two years before being cleaned

up, it being generally the practice in the monthly clean up to take up only the upper portions of the sluice. The period will of course be governed by the extent of the operations.

Operating the Sluice Box.—The maximum size of material that should be allowed to pass into the sluice box will vary according to the nature of the material being treated. The most difficult material to sluice is a generally fine heavy alluvium containing very little stone. It requires more water and more gradient. The presence of a more or less regular quantity of stones considerably increases the efficiency of the boxes both in recovery and quantity treated. The presence of the stone causes a certain "liveliness" or "quickness" in the finer material which permits the minerals of higher specific gravity to settle with greater ease. The stones, in other words, prevent the fines from packing.

In tin recovery the more common method of operating the sluice box is by what are known as open riffles, i.e. stops placed across the sluice box at intervals varying from 12 ft. to 20 ft. and made of 3 in. by 2 in. hardwood placed on edge. The sluice men who stand in the box operate on the different sections and continually push back the alluvium against the flow of the water by means of a shovel. This allows the heavier particles to settle and the waste to flow away. Several men are required for the work. They usually operate in groups behind each other and move from the tail of the box towards the head. Sometimes in dealing with richer material special men are kept continuously near the head of the box aided by a gang that rove up and down the box. The efficiency of the work depends to a large extent on the attention of the sluice-man to his duties, and in a well-managed "box" 80% of the tin ore should be recovered in the first 30 or 40 ft. of the sluice.

Towards the end of the box it is not unusual for the men to operate with sluice forks of 10 to 12 "tynes",[1] instead of shovels (Fig. 56). The fork is buried in the material at a flat angle and the handle gradually brought to a nearly vertical position, while at the same time a lateral swinging motion is given. Here again the object is to quicken the sands and allow the cassiterite to settle.

With the object of reducing the number of sluice-men employed in the boxes, close riffles as used in gold recovery have been introduced in tin work. They consist of sections of 3 in. by 2 in. or 4 in. by 2 in. spaced 3 in. apart across the entire length of the sluice box. They produce a rippling action in the stream of material and water which allows the gold to settle into the cavities between the riffles, thus producing a similar action

[1] Tynes means teeth.

as that attained by the sluice-man shovelling the material back against the stream. They have been used with success in tin recovery.

Further economies in this regard have been attempted by the introduction of mechanical appliances such as rakes, etc., but so far they have met with little success. More recently still an attempt has been made to rapidly classify and jig the material and though success has been claimed in several cases, the authors

FIG. 56.—A SMALL SLUICE BOX—Shovels in use at the head and sluice forks lower down.

consider that the method is still in the experimental stages and it is premature to put it forward alone as an effective method of treatment. The Day Rocking Concentrator (see *The Mining Magazine*, March, 1928) is the latest addition to what might be called mechanical sluice boxes and introduces new features for agitating or keeping the mass in the box " alive " and in a condition favourable for the settlement of the tin-ore or gold. It possesses a simplicity beyond the jig and certainly has advantages over some of the types of mechanical agitators that the authors have observed,

particularly in the clear and unobstructed passage of the material along the sluice.

With the open riffle method the sluice-man periodically adds a riffle from time to time as he notes the gradual increase in the value of the material retained in the sluice. This goes on until he has a more or less full sluice box. The supply of material coming to the sluice is then shut off, and he begins what is then called "running down" his sluice. Clean water only is passed through the box, and the sluice-men continue to shovel the material up against the current. Occasionally a riffle in each series is removed and the shovelling process continues and becomes

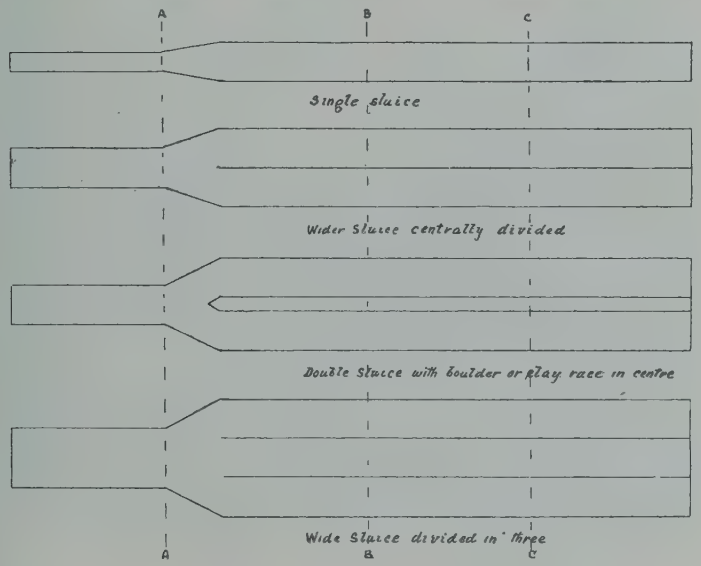

Fig. 57.—Showing Various Types of Sluice Boxes.

more vigorous as the condition of concentration increases. This shows first of all near the head of the sluice and from here the higher grade concentrate is from time to time removed to an adjacent streaming box and the dressing continued until the concentrate has attained a value varying from 65 to 70% cassiterite. The "running down" process continues until all riffles have been removed (the riffle at the tail of the box is generally retained), and the concentrated material is then shovelled up stream against the current until it is considered sufficiently prepared to pass to the streaming box for final dressing. In large plants this "running down" process usually takes place monthly.

In operating with the closed type of riffle it is possible each day to remove the rough concentrates from near the head of the sluice and pass them to the dressing plant for final treatment. Under this system, ore is being continuously recovered and shipped and many favour it in preference to the open riffle system wherein a longer period is lost each month in the " clean up " of the whole of the material from the sluice.

Wide sluice boxes are sometimes divided down the centre with the dual object of regulating the flow of the material and affording a means of " cleaning-up " one-half of the sluice whilst

Fig. 58.—A Graduated Sluice on a Tasmanian Mine. The elevated boxes are narrower and catch the coarser tin ore. The wide and shallow sluices in the foreground deal with the fine material.

the other half continues to operate. The centre division sometimes takes the form of a narrow sluice, particularly where large boulders or lumps of clay material occur. The sluice-man at the head of the box forks these into the boulder sluice, as it is sometimes called, and they pass to the tail of the sluice without further impeding the working of the sluice.

Three divisions are also not an uncommon feature in very wide sluice boxes, one of which can be " cleaned-up " whilst two operate. This is favoured when the tin concentrate is very fine, as it does not so greatly reduce the normal working width

of the sluice. It only means a 33% reduction against 50% in the case of the single division (Fig. 57).

Rarely are under-current devices (as generally understood) used in connection with tin recovery work. Sometimes, however, when the alluvium contains both coarse and fine tin the material is first passed through a narrower type of sluice designed to save only the coarser tin. At the end of this sluice the "tails" containing the fine tin are discharged over a grizzly of such dimensions as will eliminate all stone above 1 in. size. The fine material which passes the grizzly is fed to a series of wider boxes where the finer tin is recovered.

FIG. 59.—ANOTHER VIEW OF THE ARRANGEMENT SHOWN IN FIG. 58.

The object of the wider box is to allow the material to pass through in as thin a layer as possible and thus afford opportunity for the fine tin to settle. It can be readily understood that the deeper the flow through the box the less chance there is of the fine tin settling, particularly in the case of "open riffles". With an arrangement of this nature some provision should be made for adding additional water at the head of the fine sluice box, as the absence of the larger stones tends to allow the material to pack and makes its removal more difficult. Additional water or increase in gradient will assist in overcoming this. The illustrations in Figs. 58 and 59 show an installation of this type on a small mine in Tasmania.

Cleaning the Crude Concentrates.—After the concentrates have been removed from the sluice box they are more or less in a crude condition and have to be cleaned and prepared to market standard, which is generally accepted as being equal to 70% metallic tin. The cleaning operation is usually first done in a small streaming box (a small sluice box) about 8 ft. long, 2 ft. wide, and 1 ft. deep on a 5% grade. The method is a continuation of the process of " running down " in the main sluice box and the tails from the process are usually run into a pit and saved for re-treatment. The " heads " from this operation are generally of 70% quality and ready to bag at once. The middlings and tails are usually passed to a Willoughby tin dresser for further refinement (Fig. 60).

Fig. 60.—A Group of Willoughby Tin Dressing Machines in Nigeria.

The Willoughby consists (in the main) of a cylindrical vessel with a screen at the bottom through which it is possible to pass a rising stream of water under a gradually decreasing head. The crude concentrates are placed in the cylinder and the water turned on. As the water begins to rise through them they are given a rough swirling motion and then allowed to settle against the upward current. The dross, consisting generally of silica, black sand (ilmenite, magnetite, etc.), is brought to the surface and after due

time is allowed for settlement is scraped off with a thin flat scraper. The process is repeated until a clean concentrate remains (Fig. 61).

Where large quantities of low grade concentrates have to be dealt with, it is advisable to instal a magnetic separator for the work. These machines operate with conspicuous success and are gaining well-earned popularity. They are an accepted portion of the equipment in all modern alluvial tin-dressing plants.

In the Gaia Valley, Portugal, where the Portuguese American Tin Company are operating, the concentrate from the jigs averages

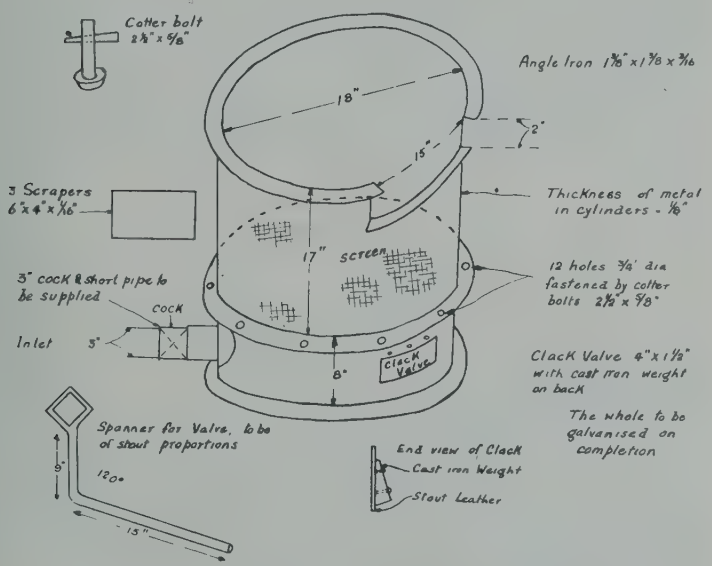

Fig. 61.—A Type of Willoughby Tin Dresser.

about 30% tin, the remainder being magnetite and ilmenite. This concentrate is dried and then passed through a Wetherell magnetic separator having four magnets; these proved to be on the weak side and the tin content of the dressed mineral could only be brought up to about 68%. The addition of another and stronger magnet made it possible to produce a finished concentrate which averaged over 70%. This was a satisfactory grade. The sun's heat would do most of the drying of the concentrates, but at times during the cold season artificial heat was used. This was somewhat expensive, as fuel in Portugal is very limited in that locality.

At the Ropp Tin, Ltd., Nigeria, the rough concentrates from the sluices pass to Harz jigs on the dredge, then into metal drums, and are transported to the main dressing plant where they are passed through a sizing trommel into Harz jigs, the tailings and middlings going to Wilfley tables ; some of the finished product is bagged from these and some passes through an Ulrich magnetic separator, then to storage bin for shipment. This is a somewhat complicated treatment due to the metal content and character of concentrate not being the same over the whole of the property.

These instances are merely given as practice in two fields some distance apart, but such plants may be varied to suit the work in hand.

HYDRAULIC ELEVATING.

The first essential to hydraulic elevating is a plentiful supply of water at a considerable elevation above the deposit to be worked. Under ideal conditions it may be possible to attain this with a short head-race (leat or ditch) and a pipeline of no considerable length to conduct it to the work. In less favoured circumstances it may be necessary to conduct the water in open channels for many miles and even then have a considerable length of pipe connecting penstock and paddock. Dams, flumes and bridges may have to be built and before any decision is made the engineer-in-charge should conduct most careful enquiries.

Jet elevating constitutes one of the cheapest forms of mining, but balanced judgment is needed in making a decision. The capital cost and heavy maintenance charges on a long ditch line and the low efficiency of the machines themselves might easily outweigh all other advantages and justify the adoption of some alternative method of exploitation.

Height to which Material can be Lifted.—The height to which material can be lifted depends upon the pressure (hydrostatic head) of the elevating jet and in a plant of good design the machines will do efficient work in lifting a height of 10 ft. to 15 ft. for every 100 ft. of hydrostatic head, and in operating in ordinary alluvial gravels they will lift 6 to 7 cubic yards per hour for every sluice-head of water used in the jet (Fig. 62). In other words, for every cubic foot of gravel-drift lifted a maximum height of 15 ft. two cubic feet of water issuing from the jet under a head of 100 ft. is required. From this the following formula has been deduced by H. A. Gordon :—

FIG. 62.—A HIGH ELEVATOR LIFT.

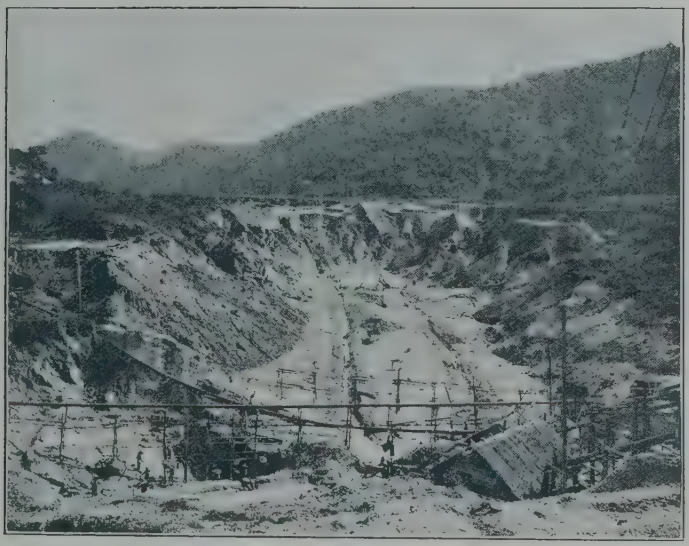

FIG. 63.—AN EXAMPLE OF A GOOD LAY-OUT OF FEED RACES TO DELIVER DISINTEGRATED MATERIAL FROM THE WORKING FACE TO THE ELEVATOR SUMP.

Let x = The quantity of material lifted per hour;
n = The number of sluice-heads used;
H = The hydrostatic head in feet at the jet;
C = A coefficient, namely H, in hundreds of feet multiplied by 15

$$x = \frac{H \times n}{C}$$

For example, suppose that 6 sluice-heads (360 cubic feet per minute) of water are issuing from the jet under a head of 300 ft., what quantity of material would it lift and to what height?

Fig. 64.—Another Example as in Fig. 63.

$\therefore$ $3 \times 15 = 45$ ft., the efficient working height of the elevator.

Then $\frac{300 \times 6}{45} = 40$ cubic yards per hour the quantity of material the elevator will lift assuming it is of reasonably good design.

The quantity handled, naturally, will depend to some extent on the nature of the gravel and the efficiency of the lay-out in the paddock for facilitating the delivery of the material to the elevator sump. In other words the paddock should have sufficient feed-races of ample proportions to conduct the broken material from

the working faces to the elevator pit (Figs. 63–5). The depth of the elevator sump should be such as will allow of ample gradient being maintained in the feed races as the work in the paddock extends, otherwise frequent removal of the elevator position becomes necessary. To some extent the nature of the bedrock is a limiting or controlling feature. In any case there is an economic ratio between height of lift, the depth of the sump, and the maximum distance of the working face from the elevator, Every deposit is a problem in itself and the care with which the engineer in charge watches the varied conditions does much in making exploitation successful.

FIG. 65.—ANOTHER EXAMPLE AS IN FIGS. 63 AND 64.

A point of particular importance to note is the distance between the elevating jet and the throat of the elevator. If this can be varied, as in the Heinecke machine, it will be observed that it has a marked effect on the lifting power of the water and a slight amount of experimental work will determine the position of maximum efficiency.

Elevating in Stony Ground.—In circumstances where the ground is so stony that elevators are not ordinarily applicable owing to the presence of a quantity of stone of just such a size

as to be capable of choking the throat, and where even large diameter pumps cannot be used, modified forms of the elevator have proved most successful accessories.

Fig. 66 shows a design where the material from the feed-races first passes over a grizzly before dropping into the elevator sump. The undersize is elevated to the sluice boxes whilst the oversize is cast on to a chute, or large pipe, supplied with a jet, and driven upward and backward into some worked-out section of the paddock. Where ample water is available such devices are most efficient for stacking boulders and are valuable labour savers. Only the larger boulders should be so eliminated. The smaller stones should go to the sluice boxes as they play a valuable part in disintegrating sticky material and in maintaining in the sluice boxes that liveliness or quickness in the sands which allows

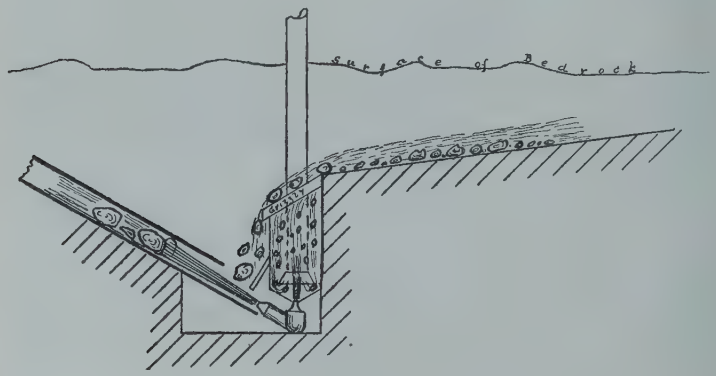

Fig. 66.—A Device for Elevating Boulders.

the heavier and valuable products to settle and concentrate near the bottom of the boxes.

Useful Effect of Hydraulic Elevators.—A most useful series of observations to determine the efficiency of their elevators was made by the Blue Spur Consolidated Co., of Otago, New Zealand, where three elevators were operated. The length of the supply pipe and head were accurately measured. The results are summarized by Gordon as follows :—

No. 1 *Elevator.* Lifting 66 ft. with a jet $3\frac{1}{2}$ in. diameter and a head of 445 ft. The supply pipe was 15 in. diam. and 1,740 ft. long. The sluicing nozzle was 4 in. diam. under a head of 400·5 ft. and the supply pipe 11 in. diam. and 1,580 ft. long.

The discharge from a $3\frac{1}{2}$ in. jet with 445 ft. head is equal to 9·35 cu. ft. per sec. and from a 4 in. jet under 400·5 ft. head is

13·1 cu. ft. per sec. The loss of head due to friction in the 15 in. pipe would be $\frac{p\,l\,c\,v^2}{2g\,a}$ = 1·5 ft. per 100 ft. × 17·4 (length of pipes in 100's of feet) = 26·1 loss of head, and in the nozzle the loss would be 13·4 ft. per 100 ft. of length, which × by 15·8 (the length of the line in 100's of feet) = 212 ft., *i.e.* total loss of head.[1]

The actual discharge under the hydraulic head would be: Jet for elevating 445 − 26 = 419 ft., giving a discharge of 9·1 cu. ft. Nozzle for sluicing 400·5 − 212 = 187·5 ft., giving a discharge of 9·1 cu. ft. The elevator therefore is lifting 18·2 cu. ft. of water per second, exclusive of the gravel, to a height of 66 ft. with 9·1 cu. ft. of water under a hydraulic head of 419 ft. Whence $\frac{100 \times 18{\cdot}2 \times 66}{419 \times 9{\cdot}1}$ = 31·5% of the actual power employed to lift the water and material.

No. 2 *Elevator.* Lifting 62·4 ft. with a jet 3¼ in. diam. under a head of 400·5 ft.; the supply pipe being 15 in. diam. and 1,580 ft. long; the sluicing nozzle being the same as used for No. 1 Elevator.

By using the same calculation as shown in No. 1 the loss of head in the 15 in. pipe is 20·6. The hydraulic head is therefore 400·5 − 20·6 = 379·9 ft. The discharge of the jet under this head is about 8·64 cu. ft. and the discharge from the sluicing nozzle the same as before—namely 9·1 cu. ft. Therefore 8·64 cu. ft. coming from the jet per sec. under a head of 379·9 ft. lifts 17·74 cu. ft. of water, exclusive of material, to a height of 62·4 ft. per sec. Therefore $\frac{100 \times 62{\cdot}4 \times 17{\cdot}74}{379{\cdot}9 \times 8{\cdot}64}$ = 33·7%.

No. 3 *Elevator.* Lifting 42 ft., the jet being 2⅝ in. diam. under a head of 354 ft.; the supply pipe 15 in. diam. and 1,333 ft. long; the sluicing nozzle being the same at each elevator, as previously stated.

The loss in head due to friction in the supply pipe 15 in. diam. is 7 ft. Therefore 354 − 7 = 347 ft. hydraulic head, which gives the jet a discharge of 5·38 cu. ft. per sec. Therefore the water from this jet, under a head of 347 ft. lifts 14·48 cu. ft. of water to a height of 42 ft. per sec. Therefore $\frac{100 \times 42 \times 14{\cdot}48}{347 \times 5{\cdot}38}$ = 32·5%.

It may therefore be said that these hydraulic elevators have a useful effect of about 35% of the power employed to do the work and will lift from 50 to 60 cu. yds. of gravel per hour. This quantity of gravel per hour is equal to 0·45 cu. ft. of gravel per sec. and, assuming that gravel is two and a half times heavier than

[1] See formula for friction of water in pipes.

water, this would be equal to 1·125 cu. ft. of water lifted. If we take the last calculation and add this to 14·48, it will give the actual percentage of the power used, $14{\cdot}48 + 1{\cdot}125 = 15{\cdot}605$ cu. ft., and $\frac{100 \times 42 \times 15{\cdot}605}{347 \times 5{\cdot}38} = 35\%$ of the power employed. From these experiments the following formula can be deduced :—

Let Q = the number of cu. ft. of water per sec. used in the elevating jet ;

q = The quantity of cu. ft. of water per sec. in the sluicing nozzle ;

H = The hydraulic head of water on jet in ft. ;

h = The height to which the material is lifted ;

N = The quantity of material lifted in cu. ft. per sec. $\times$ 2·5 ;

E = The efficiency of the elevator ;

$$E = 0{\cdot}35 \frac{h \times (Q + q + N)}{H \times Q}.$$

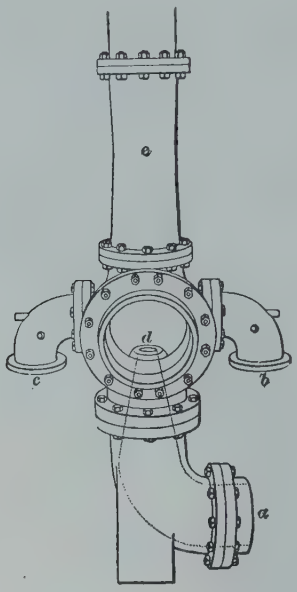

Fig. 67.—The Evans Elevator.

Types of Hydraulic Elevators. — Hydraulic elevators as commonly used are of two types, viz. submerged jets (placed in a sump into which the broken gravel is sluiced) and suction jets

(furnished with a suction pipe led to the material to be raised). Suction jets are sometimes fitted with suction pipes fully 100 yards long and at that distance work satisfactorily provided the height of lift *in the suction pipe* is low, this condition being due to the

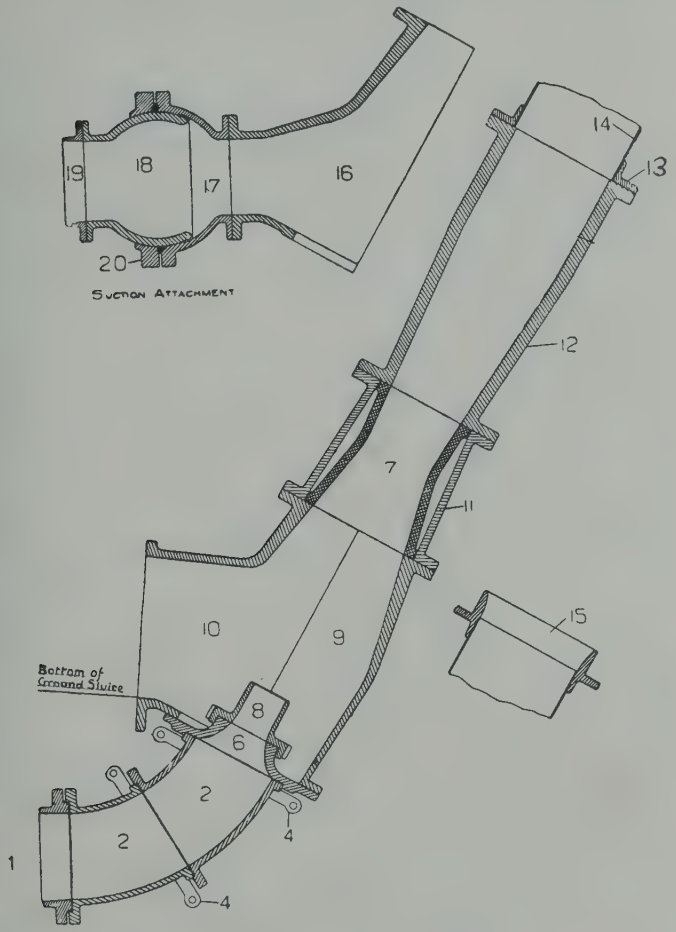

Fig. 68.—The Hendy Elevator.

considerable friction of the gravel, sand, etc., accompanying the water in the pipe, to move which solids the velocity of the water needs to be as high as possible. Output with all types depends wholly on the speed of the water. The great disadvantage of the

suction type is the delay caused when chokage occurs and the arrangements which must be made to obviate it. For this reason the submerged jet as a type seems to hold favour. Types such as the Evans and the Hendy have been so designed that they can be used in either form by merely altering sundry attachments. They are shown respectively in Figs. 67 and 68.

It is commonly known that the flow through the suction pipe is due to a vacuum having been formed by the jet of water at the base of the uptake or delivery pipe and also that when reached this jet forces the material up the pipe named. On the other hand few operators recognize and appreciate the effect of this vacuum on the efficiency of the elevator. It is a commonly overlooked fact that the use of a suction pipe makes scarcely any material difference to the lifting power of the elevator. Put in other words, most elevators will lift from the jet upwards equally high whether they have or have not a suction pipe attached. This, for the reason that a vacuum is formed in every type of elevator—it is impossible to prevent it. It is of course apparent and utilized in the suction type, but its existence is hidden and not used in the case of the submerged type and unsuspected ordinarily when the elevator is not being "fed" at all; nevertheless its production is the principal cause of lack of efficiency in this device, as much of the energy of the jet is thus absorbed. However, as this vacuum is unpreventable, the next best thing is to turn it to account and make it return some of the power that its production has absorbed. This is most easily practicable by so attaching a pipe that air may enter freely and continuously to the fullest extent at the right place and then there is a hydro-pneumatic lift within the uptake.

Air lifts, pure and simple, are now receiving considerable attention. Theoretically there is no limit to the height to which they will lift, but in practice the length of pipe (uptake) below the surface of the water being lifted must be about twice the height from the same surface to the point of discharge. This deep submergence necessitates a costly and capacious air compressor and a deep well, and for that reason greatly retards their commercial application.

In the hydro-pneumatic elevator the air enters automatically and free of cost and appliances, and its presence within the uptake is wholly advantageous. Apart from its lifting power it reduces the friction in the uptake and makes the material in the column a springy, buoyant, and yielding mass with an upward tendency of its own, thus unloading the jet of much of its burden, particularly as the column is now reduced in weight (being part water and part air) so that a lower pressure of water is permissible or a higher lift with existing pressure, or again a greater velocity and hence a greater output. It is to be noted, however, that to obtain the fullest

advantage from the air the uptake pipe must be vertical, which, however, is true in any case, as it makes the shortest journey between any two levels. When out of the vertical the solids drag their upward way against the resistance of the inclined bottom, thus losing the advantage they otherwise possess of having lost some of their weight through being wholly immersed in water.

So far as the authors are aware, the "Heinecke" improved jet elevator (Figs. 69 and 69a) is the only machine which embodies this hydro-pneumatic principle, and in one of the earlier types they attained a lifting ratio of 1 : 4·65. The hydrostatic head at the elevator tip was 67·13 ft. and the height lifted was 14·41 ft. In actual running the machine did 12 yds. per hour. Over the whole period of the test (4 months) including clean-up and all stoppages the capacity was 7·6 cu. yd. per hour. The ground was particularly free of large boulders. The diameter of the bit was $2\frac{1}{2}$ in. and the throat was 4 in. The elevator consumed 2·0966 cu. ft. of water per second and the monitor consumed 1·2679 cu. ft. of water per second.

Without any such arrangement for the admission of air as described above, the elevator will not lift the same quantity of water and material if the water rises in the sump hole above the mouth of the elevator and thus excludes the air. Robertson in New Zealand patented an appliance to remedy this and led a supply of air into a hollow chamber surrounding the jet by means of a 1 in. pipe, the upper end of which stood at such a height as would reach above the level of the water in the paddock (Fig. 70). The advantage of this, as also in the case of the Heinecke, is that in the event of flooding in an enclosed paddock the elevators (so long as the valve controlling the water supply is above submergable level) can be started up at once and will operate as pumps to unwater the paddock and work quite efficiently though entirely submerged.

The simplicity of jet elevators, their low capital cost, and the ease with which they can be moved are the principal factors which have established their utility.

Hydraulicking Practice.—The best practice is to move all the bank through the sluices at one operation. By so doing, the fine and coarse material is in the proper proportion, so that if for any reason the supply of water is cut off, thus leaving the sluices loaded, then when the water is turned on, the material will begin to move, as the flow of water among the boulders quickly starts cutting the fines from under the coarse, and soon the sluice is clear. It looks to the beginner that it must be good practice to run off the top soil and fine material first, then clean up the bottom material, but it is not. When the heavier coarse wash is being moved alone, if for any cause the water slows up or stops, it is a tremendous task to clean out the sluices ready for fresh material.

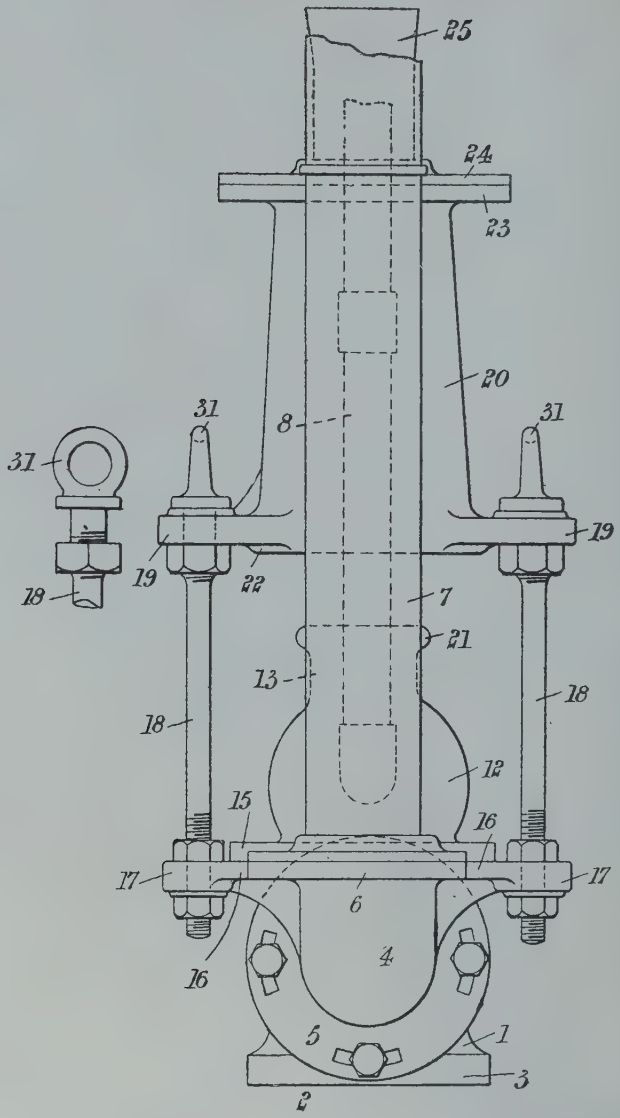

Fig. 69.—The Heinecke Improved Jet Elevator (side elevation).

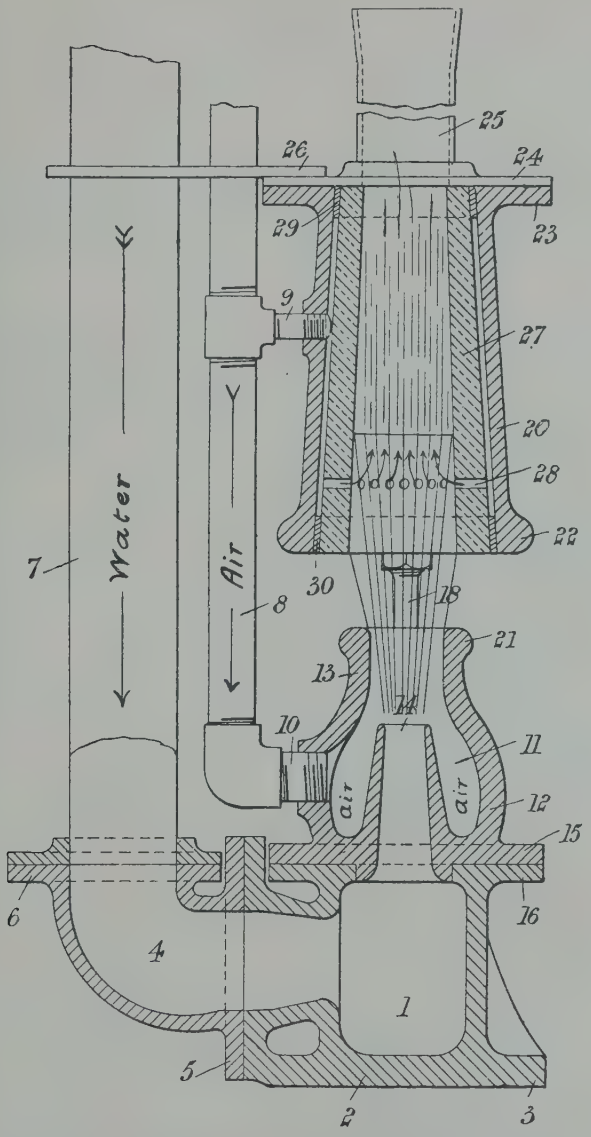

FIG. 69A.—THE HEINECKE JET ELEVATOR (CROSS SECTION).

SURVEY OF GROUND WORKED.

The maintenance of a record of the work accomplished is an important item in all mining operations and the manager should take adequate steps to see that this is done on a scale proportionate with the work in hand.

In alluvial mining operations are often carried on in several working faces simultaneously, and these should be measured regularly every month to ascertain with reasonable exactitude the cubic yardage of material excavated. Readings should be

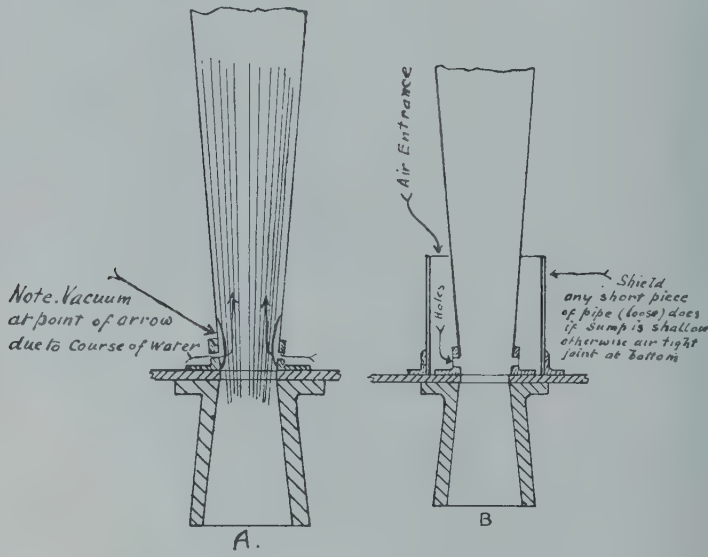
Note. Vacuum at point of arrow due to Course of water
A.
Air Entrance
Holes
Shield any short piece of pipe (loose) does if Sump is shallow otherwise air tight joint at bottom
B

FIG. 70.—ILLUSTRATING THE ADVANTAGE OF AN AIR INLET TO ELEVATORS.

recorded of the depth of ground from surface to bedrock. This should be done weekly and at regular intervals along the edge of the working face, so as to determine the average depth of the ground worked during the month, while from several survey stations established well back from the working faces and in positions not likely to be disturbed for many months a more or less permanent base should be established. From these stations tacheometer readings of bearing and distance are taken to the various points from which the depths were last recorded.

This affords the information necessary to determine the difference between the old and the new line of the working face. A planimeter run around the two outlines will give the superficial

area of the worked ground. Knowing this, and the average depth, very reasonably accurate results can be obtained and without recourse to more elaborate methods.

DISPOSAL OF TAILINGS.

The disposal of the waste from alluvial mining operations and dredging is often a very vexed question, particularly in agricultural districts where operations in many cases have been restricted by legislation. Costly litigation has followed the pollution of rivers by sludge from mining work, and dredging

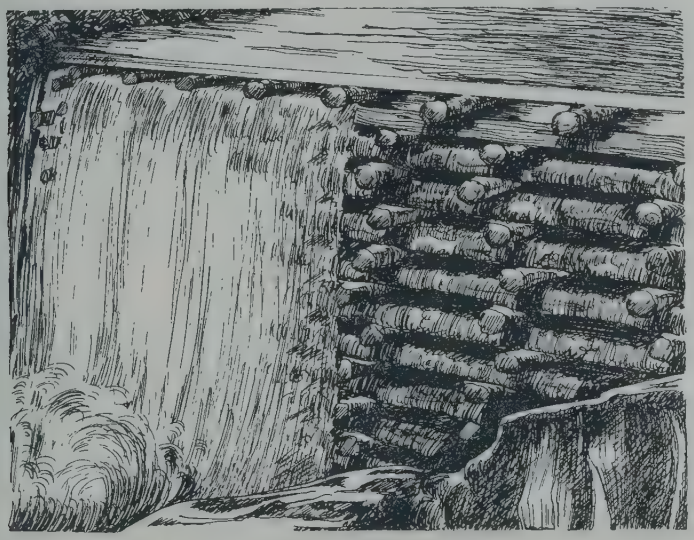

Fig. 71.—A Log Crib Dam for Retaining Tailings.

companies have been called upon to re-soil dredged lands. This aspect of the problem should not be lost sight of by engineers examining land in settled districts. Apart from this, the prodigal delivery of tailings into a stream may cover up other valuable mining land or so pollute the water as to render it unfit for hydraulic purposes.

At the inception of operations it may be necessary to retain tailings only so long as to allow of an old working place being used as a dump. Often and particularly where water is scarce they are retained in such form as to constitute temporary dams wherein

the water is allowed to settle and returned for re-use. Gravel pumping problems commonly call for this economy in water. The type of retaining dam for this purpose will vary according to local conditions. Where timber is plentiful a log crib dam as in Fig. 71 will meet the conditions. The dam is not necessarily water tight; it is merely a retaining medium to hold up the tailings and allow the surplus water to pass away. In districts where timber is scarce a very serviceable dam can be constructed by using wire netting in special form. The wire netting is worked up into cylinders by lacing the side with wire. These are then filled on the site with stone and the ends fastened before being placed in position. A sand and boulder backing is given and the next layer of "netted baskets" placed across the site, gradually rising in step fashion as shown in the sketch Fig. 72. Such a dam can be increased in height from time to time to meet further demands for storage. Overflow takes place over the crest of the dam, erosion being prevented by the netting baskets.

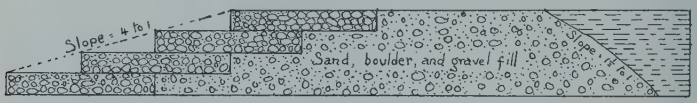

Fig. 72.—A Wire Netting Dam.

Such arrangements as those described above are operated under natural gravity conditions. With less favourable conditions it may be necessary to stack the tailings, in which case this may be done by the aid of an hydraulic elevator, a bucket elevator, or a conveyor belt. These vary over a wide range from simple apparatus to large installations involving big capital outlay.

For further reference to this subject see pp. 140–142.

EXAMPLES OF PRACTICE.

The Pioneer Tin Mine.

Description.[1]—The Pioneer Tin Mine is situated on the ancient channel of the Wyniford River which, with the Cascade River at Derby (Briseis Mine), Main Creek, Weld River, O.K. Creek, and other minor tributaries of the Ringarooma River, formed part of the original river system of the north-east of Tasmania. The width of ground worked is about 1,200 ft. and varies from 70 to 130 ft. in depth. There is no overburden or any heavy wash, the ground consisting of sand or drift with seams of waterworn pebbles and pug, derived from the denudation of the granite ranges, through which the Wyniford River runs. The working faces carry tin from the

[1] For these notes the authors are indebted to Mr. Cecil Ryan, the general manager of the Pioneer Tin Mining Co., Ltd., Tasmania.

surface down, the upper drifts being cemented for a depth of from 10 to 40 feet and, after being caved with the nozzles, have to be broken with explosives and further reduced by hand breaking to a size that will enable the lumps to travel through the feed sluices, subsequent disintegration being completed by their passage through the gravel pumps and tail races. Owing to the width and depth of tin deposit, and to its being below the gravitation level, the whole of the ground has to be elevated.

Method of Working.—The method of working is as follows :— Two electrically driven centrifugal sluicing plants carrying 15 in. gravel pumps and situated one on each side of the lead, pump the drift and sluicing water from the nozzles a vertical height of 98 and 106 ft. respectively into the tail races built on the north and south banks of the lead. The tailings are discharged into a pit at the end of the tail race, from which they are lifted on to an endless 24 in. conveyor belt which stacks them on the tailings dump, the sluicing water overflowing from the pit which is set at the drainage level. The tail races are cleaned up every six weeks, the tinstone, which is of medium fineness, being taken out fairly clean and finally dressed in the streaming boxes to 73·5%. The residues carry monazite, zircon, ilmenite and wolfram.

Plant.—The Company develops its own hydro-electric power. From a rockfill dam with concrete facing 57 ft. high, on the Frome River, into the watershed of which another stream is diverted, the water flows 1½ miles to the head of the power station penstock, from which it issues to the double nozzle impulse wheels under a head of 435 ft. The spent water from the wheels is brought 5 miles through a race and inverted siphons to a hill at the back of the mine and 300 ft. above it, from which it is piped down to the nozzles in the working faces.

The power station plant comprises three 325 k.w., 3 phase, 50 cycle A.E.G. generators driven by 3 Voith impulse wheels (oil-pressure governors) current being generated at 6,600 volts and transmitted 5 miles to the mine through hard-drawn bare copper cable 50 sq. m.m. section on wooden poles. The main transmission line terminates at a mast switch opposite each sluicing plant in the mine, from which the current at 6,000 volts is carried through an armoured cable to the switchboard in the sluicing plant without transforming. At each mast switch is a small substation carrying lightning arrester gear, H.T. switch cases and fuses, and transformers for reducing the voltages to 415 v. for the motors driving the tailing conveyors. Tirrill automatic voltage regulators at the power station keep the voltage constant, independent of the load.

The power station staff consist of 3, one electrician at £7 and 2 assistant drivers at £5 8s. each per week. Generating costs

vary from £2 to £2 10s. per h.p. per annum. Normally two generating units are run in parallel, one unit being spare. All electrical repairs, including re-winding of motors and generators are carried out by the power station staff.

Sluicing Plant.—Equipment comprises two barges carrying 15 in. diameter gravel pumps in series, rope driven by a 350 b.h.p. 6,000 volt, 3 phase, synchronous motor, switchboard carrying the usual instruments, and liquid controller which gives a speed reduction of 15% and acts as a starter, as well as small transformers for reducing voltage to 100 for barge and face lights. The gravel pumps have 3 ported runners and the design of the pumps has been evolved to meet the severe conditions of pumping heavy and abrasive sand to a vertical height of over 100 ft. Prior to compounding the gravel pumps, single pumps were used, but it was found with such high lifts that the efficiency fell off rapidly due to the wear, and the mechanical life of the pump liners was only six weeks. Since putting the pumps in series and halving the lift for each, the efficiency is maintained over a much longer period and the life of the liners increased from 6 to 15 weeks.

Owing to the width and depth of the deposit it is worked in two benches. The upper bench being worked to a depth of about 100 ft. and kept well in advance of the lower bench which is lifted to the sluicing level of the gravel pumps with hydraulic elevators.

Sluice Boxes.—The sluice boxes are 20 ft. wide and 260 ft. long with a grade of $5\frac{1}{2}$ in. to the box (12 ft.) or 1 in 26, and are divided down the centre into two 10 ft. sluices to provide for better control of the water, and have the usual arrangement of riffles every 12 ft. The central division carries a small sluice into which the lumps of pug or cement too large to roll through the race are forked, the sluice being flushed out as required. Owing to the heavy nature of the sand, the duty of the water is low, being 25 to 1 by bulk through the tail races. As thick a feed as possible is pumped, the necessary make up water being added at the head of the tail race.

Disposal of Tailings.—The tailings flow into a pit at the end of tail races and are lifted by a chain of perforated buckets on to a 24 in. 8 ply rubber and canvas conveyor belt with 500 ft. centres set on an incline of 1 in 10. A travelling tripper with a separate belt on a boom swinging through an arc of 180° stacks the tailings laterally and builds out the dump on which the conveyor is pulled round. The lattice frame that carries the main belt is fastened to a turntable at the loading end, which is also the driving end, and the frame is carried on small travelling wheels fitted 15 ft. apart, so that the whole conveyor pivoting on the turntable can be pulled round through an arc of 180°. The maximum angle of inclination for the belt is $22\frac{1}{2}$°. The main belt, bucket chain and tripper belt are driven by separate enclosed and back-geared

motors, the main belt being 40 h.p., bucket motor 15 h.p., and tripper motor 5 h.p. The belt speed is 420 ft. per min., which is increased to 460 ft., as the belt wears thin, by altering the pinion on the lay shaft of the motors. The capacity of each conveyor is 225 tons per hour and the life of the main belt averages about 1,000 cu. yds. per ft. run of belt, which for 500 ft. centres averages from 1,000,000 to 1,200,000 cu. yds.

Each tail race has its own conveyor, which is erected on the bank, both being swung inwards as their dumps are built and thus filling up the worked out ground behind. Owing to the large stacking capacity it is only necessary to move them ahead about once every four years.

Since the present Company started operations in 1899 to the end of June, 1926, 12,825,000 cu. yds. of drift averaging 1·508 lb. per cu. yd. have been pumped and sluiced for a yield of 8,627 tons of stream tin at a total mine cost of 6·603d. per cu. yd. and £40 17s. per ton of stream tin raised. £538,287 has been distributed in dividends. The average value of the ground is now 0·75 lb. per cu. yd. The average costs per cu. yd. and per ton of stream tin to June, 1926, were as follows :—

	Per cu. yd. pence.	*Per Ton of St. Tin Pounds Sterling.*
Stores	0·329	2·035
Fuel	0·714	4·416
Electric Power	0·298	1·844
Labour on Punts	0·771	4·773
Labour Faces and Races	1·580	9·773
Tailings Dam and Conveyors	0·709	4·388
Water	0·289	1·787
Tail Races	0·085	0·529
Floating Charges	0·169	1·043
Maintenance of Plant	0·596	3·687
General Charges	1·063	6·578
	6·603	40·853

Notes: The item *fuel* disappeared in 1908 when hydro-electric power was substituted for steam power.

Maintenance of Plant covers maintenance of gravel pumps, liners, runners, and all plant on barges or punts.

For one year ending June, 1926 :—

	Per cu. yd.	*Per Ton of St. Tin.*
Stores	0·346	3·594
Electric Power	0·602	6·250
Labour on Punts	0·679	7·050
Labour Faces and Races	2·366	24·568
Tailings Conveyors	1·175	12·199
Water	0·395	4·103
Floating Charges	0·332	3·445
Maintenance Plant	1·045	10·855
General Charges	1·701	17·658
	8·641d.	£89·722

Recovery per cu. yd. = 0·834 lb.
Yardage treated = 488,200 cu. yd.

Wages paid are as follows :—

Mine Foreman	£7 per week.			Motor Attendants	£4	10	0	to	£4	7	4
Shift Bosses (2)	£5	„	„	Nozzlemen	£4	5	0				
Blacksmith	£5	1	4	Sluicemen	£4	2	0				
Rough Carpenter	£4	9	0	Labourers	£3	16	0				
Teamster	£5	0	0								

The water supply is not sufficient to carry on sluicing throughout the year and it is generally necessary to shut down for from 3 to 5 months during the summer, when most of the hands are paid off and repair work only carried on with a reduced staff. The yardage sluiced per annum therefore varies with the seasons, distance of the plants from the working faces, character of the ground sluiced and other minor factors, the maximum being 996,000 and the minimum 420,000 cu. yds. per annum –the former representing 11 months' sluicing.

Details of tailings conveyor costs per cu. yd. are as follows :—

Wages	0·434
Stores	0·124
Bucket Renewals	0·093
Main Belt (amortization)	0·350
Tripper Belt (amortization)	0·123
Electric Power	0·051
	1·175 pence.

The Briseis Tin Mine.[1]

The operations at this mine are rather unique in that the company have a working face of over 460 ft. in height, the upper half of which consists of columnar and nodular basalt overlying some 260 feet of drift which occupies the ancient channel and which has not yet been tapped in its lowest limits. (See sketch Fig. 73.)

Hydraulicking has been carried on from the inception of operations on the Cascade Creek which has been followed until it now approaches what is believed to be its junction with the ancient valley of the Ringarooma River (Ringarooma deep lead). The amount of overburden to be removed has gradually increased as the lower reaches were approached and the mining work increased in depth below the present river level until at the present time the problem of removal of the overlying material is a stupendous work. The basalt is worked in big benches and much of it is trammed to dumps, particularly the bigger stuff. The smaller material up to 15 or 20 inch diameter is transported in a special boulder flume, Fig. 74, which has a gradient of 3 ft. per chain and carries 23 cu. ft. of water per second. This delivers on to a mechanical stacker to economize the limited dumping space. The underlying drifts are attacked at the base of the working face

[1] For these notes we are indebted to Mr. Lindsay Clark, the General Manager of the Company.

with monitors (Figs. 75 and 76) and jet elevators and lifted in one or more stages to the sluice boxes which are down in the working paddock below the present river. The tailings are lifted by sand

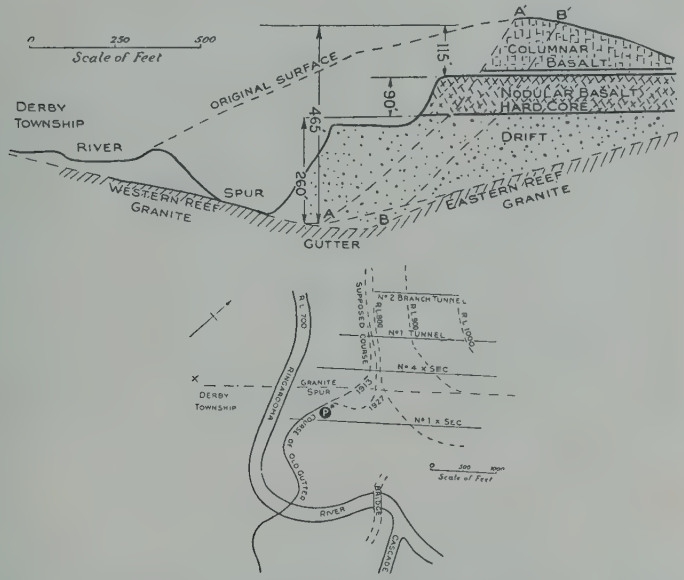

FIG. 73.—CROSS SECTION AND SKETCH MAP OF BRISEIS TIN MINE. Reproduced from *The Mining Magazine*, August 1927.

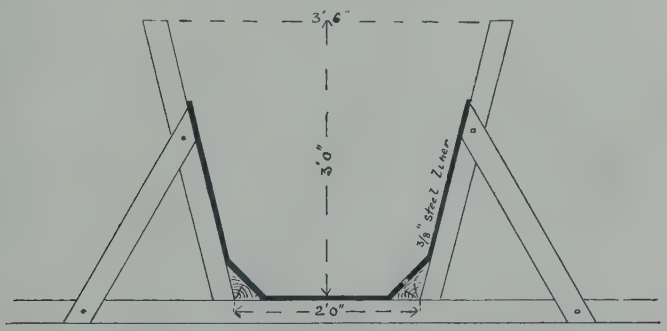

FIG. 74.—BOULDER SLUICE, BRISEIS TIN MINE.

pumps coupled to Pelton wheels and stacked above river level. The elevators have a lift of 15 to 18% of the total pressure head of the head race, and the ratio of throat to tip varies from $2\frac{1}{2}$ to

Fig. 75.—Monitors at Base of Working Face, Briseis Tin Mine.

Fig. 76.—Monitors at Base of Working Face, Briseis Tin Mine.

$2\frac{3}{4}$ to 1. The sluice boxes carry 0·6 cu. ft. of water per second for each foot of width. They are 14 and 16 feet wide and have a gradient of 1 in 22. The riffles are 16 ft. apart. A plan and elevation of the sluice and feed race is shown in Fig. 80, whilst Fig. 81 shows the tin dressing arrangements. Fig. 80 also shows

Fig. 77.—Hydraulic Elevators, Briseis Tin Mine.

the dimensions of the sluice and feed race. In the removal of the overburden a high efficiency was attained. Records show that 26 cu. yds. per sluice-head per day were removed (1 Tasmanian sluice-head = 24 cu. ft. per min.).

FIG. 78.—ELEVATED FEED RACE, BRISEIS TIN MINE. In background is the elevator shown in Fig. 77 and the tailing dump.

FIG. 79.—TIN SAVING ARRANGEMENTS (SLUICE BOX), BRISEIS TIN MINE.

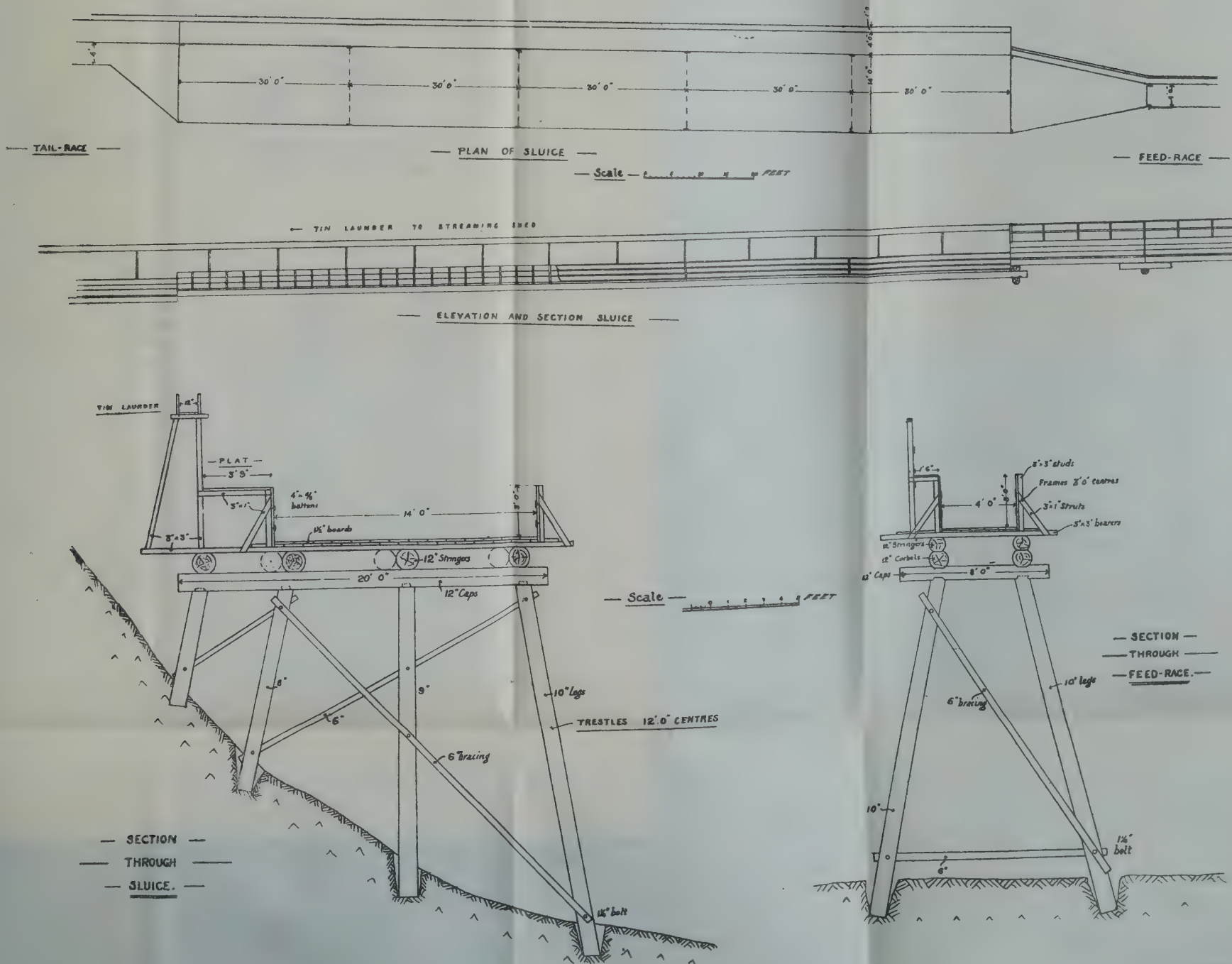

FIG. 80.—PARTICULARS OF SLUICE AND FEED RACE, BRISEIS TIN MINE.

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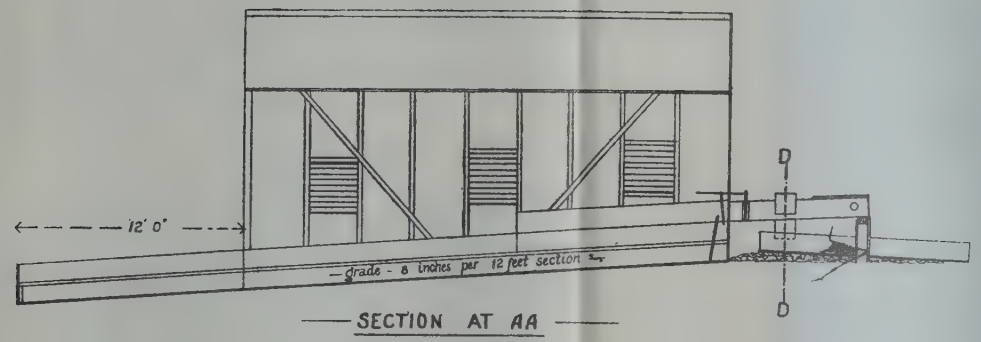
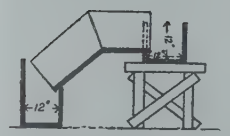


SECTION AT AA



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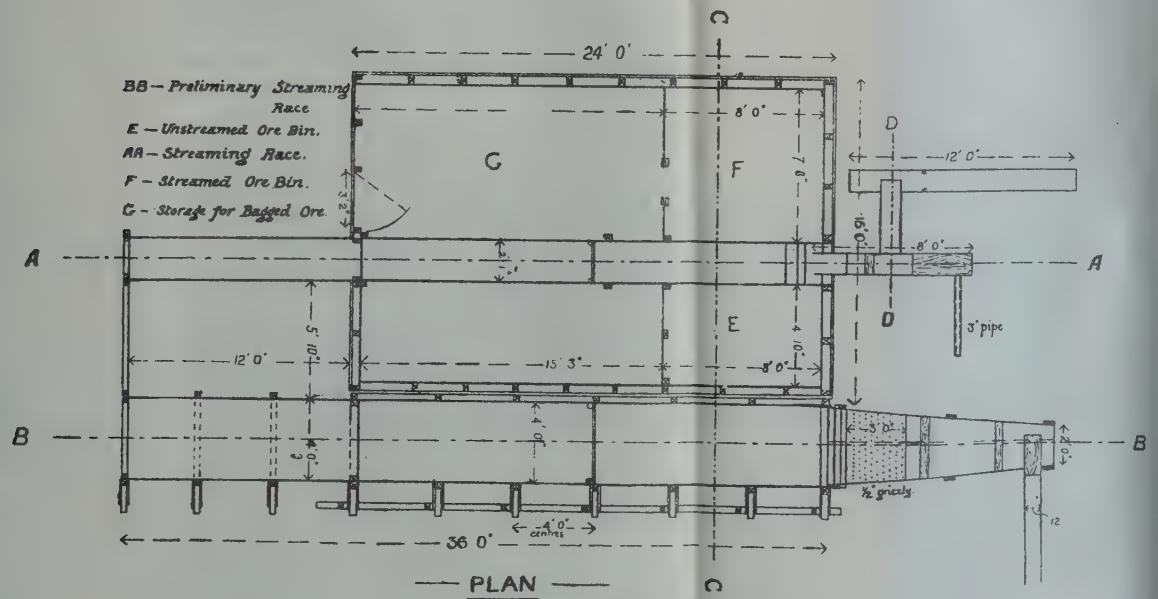
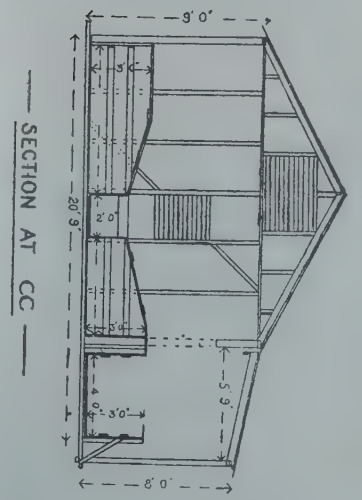
Scale 0 1 2 Feet



PLAN



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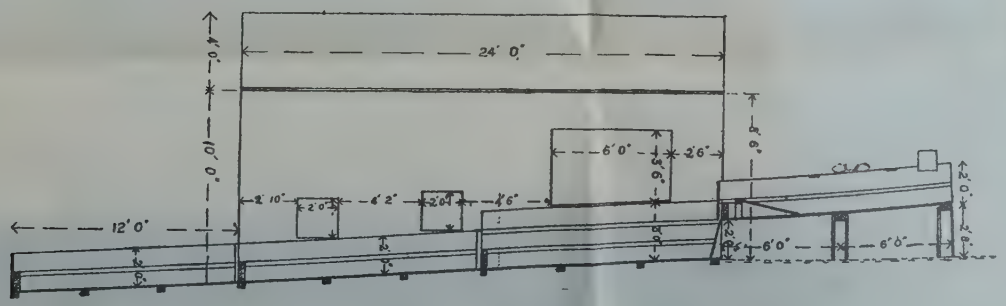

SECTION AT BB

TIN STREAMING SHED
AND
ACCESSORY PLANT

SCALE OF FEET

0 5 10

FIG. 81.—THE BRISEIS TIN MINE.

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Since these notes were prepared the Briseis Mine has been totally destroyed by floods and it is questionable if the mine can ever be re-opened and worked in the same manner as previously. It appeared to be growing beyond the limits of economic operation by hydraulic mining, and the logical method of operation would now seem to lie in drift mining from a central shaft. Remarkable efficiency was maintained up to the time of the disaster, reflecting the greatest credit on the management.

DAMS AND RESERVOIRS.

When the minimum flow of a stream is less than the daily demand for water for alluvial mining operations, or where there is a pronounced wet and dry season of the year, it becomes necessary to construct dams to conserve the excess water. Such dams are often made by erecting an embankment across a valley or at the outlet of a natural lake or basin, thus creating a reservoir. At times when the waters of a natural stream are impounded it may be necessary to consider the riparian rights of owners below and make provision accordingly. These rights are difficult to estimate and often lead to litigation, and may impose restrictions or economic conditions which would put the scheme beyond the resources of a mining company.

Dams for mining operations are seldom of a very permanent character. The life of a mine is usually comparatively short and they are consequently so designed and constructed as merely to outlive the estimated life of the mine to a reasonable extent and allow of amortization. The isolation of mining camps from the more settled centres generally precludes any consideration being given to the possible use of the impounded waters for industrial or domestic use such as power development or electric lighting and water supply schemes either concurrently or at a period later to mining operations. Of the latter types, a few instances are on record, but they are rare and generally are later adaptations in districts where mining has been carried on for a number of years and assumed some semblance of permanency.

The type of dam to be constructed is not a mere matter of selection. The nature of the mining work to be accomplished, the physical features of the site, the quantity of water to be impounded, the meteorological conditions and nature of the country are all guiding factors. The dam may be a retaining medium or it may merely be a means of directing the flow of a stream into a canal, allowing any excess to pass over the crest of the dam or weir—in other words, a diverson dam (Fig. 82).

Whatever be its destined use there are certain basic principles to be observed in its construction, the careless departure from which

invariably leads to disaster, often causing not only the loss of the dam but complete stoppage of mining operations and sometimes loss of life.

Location.—Physical conditions will often dictate the position of a dam as also the method by which a deposit must be worked and, much as one may desire to have the water in an elevated position for mining work, the flat contour of the country makes it possible only to impound it and use it by gravity and without pressure in

Fig. 82.—Diversion Dam, Northern Nigeria (Bauchi) Mine.

ordinary ground-sluicing methods. After a preliminary location had been made, accurate levels must be run to connect the working positions with the reservoir site. At the proposed site numerous borings must be made and these must be extended above and below the site as well as on the flanks. It is as well to supplement the bores by a few pits in order that the strata can be more fully observed.

Kinds of Dams. –Dams may be classified according to the material used in their construction, such as earthen dams, rubble dams, masonry or ashlar dams, wooden dams, crib dams, ferro-concrete and steel dams. There may be composite dams embodying features of two or more of the types already mentioned. The earthen embankment is the most common form of dam. It may be built on various foundations and is usually the cheapest type. When well designed and constructed it is an entirely safe and reliable structure. The embankment must be sufficiently raised above normal water level to meet the wave action caused by winds, as well as the sudden rise caused by heavy downpours. Water must never be permitted to flow over an embankment. Even rain will cause severe "fretting" and for this reason most embankments are turfed over. The by-pass or spillway should be of ample proportions and designed to carry with ease the largest estimated flood likely to pass down the stream. The entrance and sides of the by-pass should be protected from erosion by stonework or concrete. Earthen dams rarely if ever exceed a height of 100 ft. The work of construction is not, as is commonly supposed, one of very simple character which anyone can undertake. When the question of stability is taken into consideration it requires a large amount of technical skill to carry out the work.

In the construction of dams of earthwork the important point to bear in mind is that no water be permitted to percolate either through or beneath the retaining wall or embankment. Should this take place stability is at once destroyed and danger from collapse is liable at any moment. In most dams the strength of the retaining wall is entirely dependent on the weight of the material and its frictional resistance to sliding movements on the bedrock, and can be readily ascertained so long as the embankment is impervious, but should it become saturated with water, two dangerous forces are at once introduced—one tending to make it slide horizontally and the other to uplift the material. The extent of the latter force can only be determined when the point of inlet or porous stratum is known, and this is not an easy matter to determine. The lifting action is due to the depth of water above the point of entry.

Dimensions of Embankments.—Loose material when lying in a heap will on settlement assume a definite slope which varies according to its nature. This slope is known as the "angle of repose" and is less in a wetted condition than when dry. For this reason the inner or water side of an embankment is flatter than the outer slope. With coarse material this inner slope need not be more than 2 horizontal to 1 vertical, but finer material may need $2\frac{1}{2}$ or 3 to 1 and in rarer cases 4 to 1. On the outer side a slope of 2 to 1 is recommended although $1\frac{1}{2}$ to 1 is frequently used. On

high embankments benches are often placed 30 to 40 ft. apart vertically. The top of the dam should extend sufficiently above high-waterline to give a safe margin from overflow and in colder countries to protect the material exposed to water action from frost. This margin should also be sufficient to resist wave action, which on a large expanse of water assumes considerable force when strong winds occur.

Spillway or By-Wash.—The construction of a by-wash or spillway channel to carry off storm water is a most important element in the construction of a reservoir—particularly in regard to its carrying capacity and also to the site and method of construction. The channel should be cut out of the solid ground at a point as far removed from the embankment as possible and should be so designed that its cross-sectional area will be sufficient to carry the water of the largest recorded floods of the valley at a maximum depth of 2 ft. 6 in. The sides and bottom should be paved with heavy stone. Concrete on heavy wire netting forms an effective apron. Where the drop is great and the spillway cannot discharge over a natural rock-bound precipice, it should be directed over a series of steps with wells at the bottom to act as a cushion for the falling water. These wells should be of robust construction in masonry and cement or reinforced concrete for they have to withstand the pounding action of many tons of water per second. It is better to have a number of short drops not exceeding five feet rather than two or three of greater depth. Any sign of erosion in the spillway should be remedied immediately.

Offtake from Reservoirs.—The offtake from the reservoir may be by means of a tunnel or a pipe of large dimensions. The tunnel offtake is confined to reservoirs of the larger dimensions, the construction of which rarely come within the sphere of mining companies. It should be constructed in the solid ground at one side of the embankment and at a safe distance therefrom and never through the embankment, and should be lined throughout with concrete or brick set in cement. An offtake tower controls the entry of the water to the tunnel. A ladder-way in the tower gives access to the various valves. In the case of smaller reservoirs, an offtake-pipe is usually provided. This generally passes through the embankment and has concrete reinforcement to prevent the passage of water along the exterior of the pipe. The discharge of the water is controlled from the inside of the reservoir by means of balanced sluice-gates.

For further notes on dams see p. 143.

POWER SHOVELS, EXCAVATORS, AND CRANE NAVVIES.

The terms power shovel, excavator, and crane navvy have been evolved since the introduction of the original shovel, using steam power as a prime mover. With the development of electric power and the internal combustion engine, the electric motor and the oil engine have in many cases replaced the steam engine. The steam shovel, however, is still most generally used, and in places distantly removed from settlement it is either this type

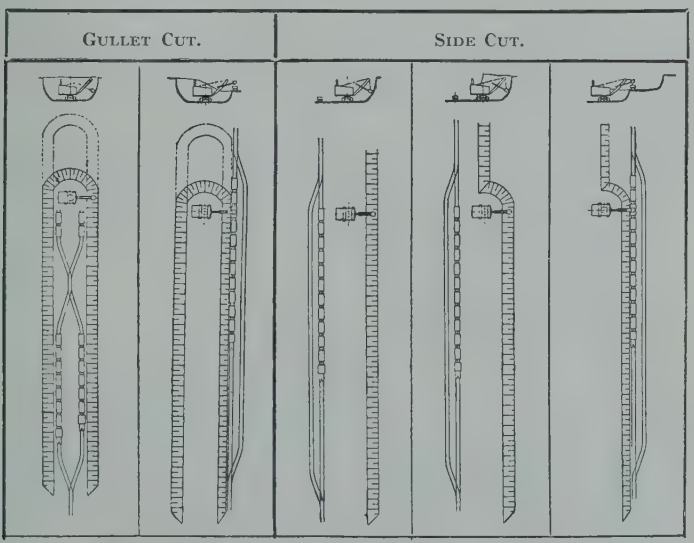

Fig. 83.—Types of Cuts in Shovel Excavation. A shows trucks on floor level behind the shovel, two tracks being provided. B shows trucks placed on track at higher level. C shows excavation in form of semi-cut, trucks being on floor level. D shows side excavation, trucks being on floor level. E shows excavation in form of semi-cut, with trucks at higher level.

or the petrol driven type which is encountered. The electrically operated type is more frequently met where progressive electrical development has taken place and where cheap power exists.

Power shovels in general may be classified according to their construction and the method of operation. Apart from the designation of the power unit such as steam shovel, electric shovel, and internal combustion (oil or petrol driven) shovel, they are classified according to their mechanical construction :—

(1) Those having their machinery on a fixed platform and capable of swinging the head of the machine on an arc of about 200°.

(2) Those with the machinery surmounting a revolving platform and capable of turning a complete circle, the centre of which is the middle of the machine.

The method of mounting the platform to attain portability varies according to the class of work to be accomplished. For railway work the machines are, naturally, mounted on trucks of standard gauge, but for other work the gauge varies from 7 to 35 feet. The wheels, two four-wheeled bogies, are small. The centre of gravity of the machine is therefore very low, and great stability is given by the wide base. This type is suitable for all classes of earthwork. Other types have the platform mounted on a caterpillar base or are equipped with broad tyred wheels.

Fig. 84.—A Shovel Driving a Gullet Cut.

Types with a Fixed Platform.—There is a general similarity in all makes of this type. The operating machinery is erected on the platform or body of the car together with the power unit. In the case of a steam plant the boiler is placed at the rear end, the engines for slewing, racking, and hoisting are near the centre and the gantry frame and crane at the front end. The platform consists of structural steel framing which is often reinforced by stout wooden members to support the steel shapes against the severe strains of operation. Heavy steel decking further strengthens the frame. A light corrugated iron building protects the machinery from the weather. On either side of the platform,

at the front end, jack braces are commonly placed. These are heavy steel brackets carrying screw-jacks and are used when

FIG. 85.—A DRAGLINE SHOVEL REMOVING OVERBURDEN.

FIG. 86.—A SHOVEL REMOVING OVERBURDEN AND A SMALLER ONE RECOVERING TIN-BEARING WASH.

the shovel is operating to prevent the machine from tipping when the crane is being swung from side to side. They are usually

designed so that they can be swung alongside the car to reduce width when the machine is travelling any distance.

As an excavating medium the power shovel made its reputation many years ago, but its most successful application to alluvial mining work may be said to have originated on the Pine Creek, Atlin, British Columbia. More recently, however, it has found much favour in Nigerian mining, where it has been more or less successful. As digging agents these machines have been undoubtedly successful, but their efficiency for mining work depends almost wholly on the magnitude of the arrangements that are made for the quick and ready removal of the spoil. In other words, no matter how economically these machines will remove overburden and excavate the valuable "wash", their success as a whole will depend on the celerity with which the excavated overburden can be delivered to the dump and the exposed "wash" removed to the sluice boxes. The ability of the machines to work at full capacity depends on the lay-out of the transport (tramming) equipment. They demand an ample trucking system of generous proportions. They are not suited to shallow deposits or frozen ground, and careful investigation is necessary before a decision is made to instal them. Both capital cost and maintenance are heavy items, particularly in remote districts.

Conditions governing the output of a steam shovel are the size of the machine, the character of the material to be excavated, the efficiency of the operating crew, climatic conditions, location of material in relation to the shovel, relation of shovel position to dumping point, and especially the efficiency of the wagon or car service. Where working under favoured conditions the maximum capacity will rarely average more than half the theoretical capacity assigned by the manufacturers. A shovel will generally be in actual operation about 40% of the working time. Time is spent in waiting for cars, delays for repairs, coaling, watering, oiling. A typical record of the log book would be as follows :—

Moving shovel . . .	10%
Breaking rock and mucking .	10%
Waiting for cars . . .	15%
Repairs	5%
Loading	60%
Total . . .	100%

The cost of excavating in clay and gravel in Nigeria per 16-hour day would approximately be as follows :—

Labour :—	1 Engineer	£2 0 0		
	1 Cranehand	1 15 0		
	1 Fireman	2 6		
	1 Watchman	1 0		
	8 Pitmen	8 0		
	1 Man hauling coal, water, etc.	3 4		
	Total per day		£4 9 10	
Supplies :—	11 Tons of coal @ £2	£22 0 0		
	Oil and Waste	12 0		
	Water	5 0		
	Fuel cost per day		£22 17 0	
General Expenses :—				
	Repairs	£2 0 0		
	Incidentals	12 0		
	Total General Expenses		£2 12 0	
			£29 18 10	

Cubic yards excavated 2,000, or 3·4d. per cu. yd.

To this must be added a depreciation charge of $12\frac{1}{2}\%$ on the capital cost of the shovel landed in Nigeria.

Fig. 83 shows various types of cuts in shovel excavations.

Figs. 84-6 show shovels in operations.

Further notes on shovels will be found on p. 112.

PART II

DREDGING IN GENERAL

Dredging.—The use of the dredge began in New Zealand. The first were of the scoop type and were mere toys as compared with the present day machines with a capacity ranging from 50,000 to 350,000 cubic yards per month, but they paid and pointed the way for the more modern mechanical equipment and the low operating costs which are so important in any industry.

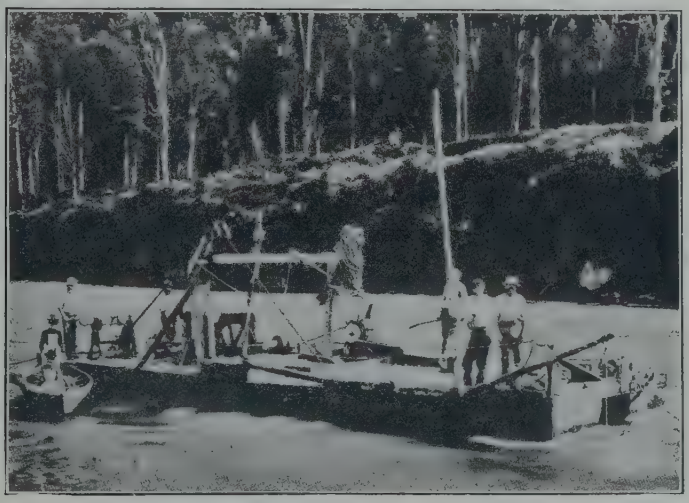

Fig. 1.—Spoon Dredge, New Zealand, 1865.

Working costs alone have been as low as $1\frac{1}{4}$d. ($2\frac{1}{2}$ cents) per cubic yard under the best conditions in California, but large scale continuous operations cost from $2\frac{1}{4}$d. ($4\frac{1}{2}$ cents) to 5d. (10 cents) for gold. Malayan tin dredging costs have been returned as low as 2d. (4 cents) per cubic yard for a season's operations. These low figures must be considered as having been obtained under ideal conditions and are not readily attainable. They do not include depreciation.

Prospecting Dredges.—There are few instances on record where justification can be shown for the installation of prospecting dredges. If there is a question of doubt, the best practice is to employ a competent, experienced engineer, and to send him out to make a report. His fee will be small as compared to the dredge cost, and he can certainly advise the investor correctly, after having carefully examined the property. Arguments advanced in favour of this method of checking a boring estimate are usually that such estimates were made in the form of a laboratory test and not on a commercial scale. In the beginning of boring—as a testing method—that statement sounded reasonable, but is absurd to-day, for boring has become standard practice and is accurate when properly carried out.

Fig. 2.—Brazilian Diamond Washing.

Diamond Dredging and Mining.—This subject can only be considered in a general way, as no successful floating dredge has been operated. The digging and elevating will not prove difficult; sizing, too, may easily be done, and then comes the rough concentration in order to reduce the bulk. Hydraulic classifiers are one of the possible means to overcome this. There must be close sizing if the classifier is to do the work at all well. Then this reduced bulk may be put over the greased tables if no better method has been proved. To attempt to reduce the bulk by the

ordinary sluice is almost impossible, for there is a very slight difference in specific gravity between the diamond and ordinary sand and gravel.

In the Brazilian ground-sluice method the sides and bottom are made from flat stones fitted roughly and packed with clay in the joints. In construction they are about 2 ft. in width and have sections of some 4 ft. in length, almost horizontal, then a 4 to 6 in. drop, and another 4 ft. up to 16 or 20 ft. total length. The clay joints soon form little catch basins, and in these the miner finds the rough concentrate. The practice is to clean up often in order to prevent any loss from overloading, and the primitive miners are most anxious to see if they are likely to have any recovery. These miners are very sharp-eyed and can almost at once detect a diamond in these preliminary clean-ups. As shown in Fig. 2, the final washing is done in clear water. The men prefer water about 3 ft. deep as this involves less stooping. The finishing is done in 2 wooden bowls, one about 20 in. over all, the other about 12 in. The larger is used to reduce the bulk roughly, and the small for the final cleaning. When the work is well done there is no chance for a profit in the tailing. Many small stones are thrown aside, for there is no market for them. Once a week the market is held and here the miners and diamond buyers meet and many heated arguments arise and a few torrid remarks pass. Thus, to be called a "Bandito" is an offence that can well be passed over by a shrug of the shoulders or a smile. The miner argues long and loud, but generally takes the buyer's offer.

In some districts (where the bedrock of the streams is hard and full of crevices) that are subject to torrential floods there have occurred many re-enrichments due to the floods sweeping many cubic yards over these natural riffles. The miners have small areas that each claim, and these are re-worked two or three times a year, and always produce some diamonds. This has been going on for over 100 years. Somewhat similar conditions are found in some of the diamond deposits of Venezuela, but without the annual enrichments.

Favourable Dredging Conditions.—What may be considered the best working conditions are rarely met with in actual practice. This may be one of the reasons why some engineers are on the ultra-conservative side and express the opinion that there are no really favourable conditions. Some difficulties must be expected and it is an engineer's duty to find a means to overcome them provided payable values are to be found. The same may be said of any class of mining.

Fig. 3.—On page 65 of *The Mining Magazine*, August, 1916, there is an illustrated account of the McGuire Placer Mine, Columbia, situated on a tributary of the Porce River. One of

the authors examined this a few years ago, and while from the picture there appear to be great quantities of large boulders, it could have been dredged and would have returned greater profits in less time than is possible by hydraulic elevators.

Fig. 4 shows, on the contrary, ground unsuitable for dredging.

Ideal ground is a comparatively level surface, ample water, not too large boulders, little clay, few buried timbers, ample volume, depth not too great, and fair value.

Information Required for Promoting.—A tremendous amount of time is wasted in this direction by promoters sending along a statement that " the yardage " is millions and the average value very high and " many people are conversant with these facts ". This

Fig. 3.—McGuire Placer Deposits. Example of ground that might have been dredged.

on the face of it sounds queer. In all reports a few photographs would greatly assist. To get attention from capital, reports with plans should show that systematic sampling has been done. The reports should give data tabulated, showing depth, distance from centre to centre of holes (state pits or boreholes and size of bore), factors used, values from each hole, surface conditions, if covered with vegetation, buried timbers, large boulders, character of bedrock (frozen or thawed, if in the north), clay, if land is cultivated, and mining regulations if in out-of-the-way districts. Some verification as to the reliability of the reporting engineer

should be made if he is little known. Promoting is not difficult providing there is something to offer and one goes about it in the right way.

Operating Life of Property Permissible.—All conditions must be considered. If in the far northern latitudes where producing seasons are confined to seven or eight months per year, and all key-men must be kept on the pay-roll, then it is advisable to increase the size of equipment and work out the property in from seven to ten years instead of ten to fifteen years, and have the return of the investment for other business in a shorter period,

Fig. 4.—Example of type of Ground suitable only for Gravel Pump or Hydraulic Elevator.

besides having reduced the operating cost. By a close analysis it may be determined just what this saving will be. Experience will dictate this.

The same general scheme can well be applied to some of the tropical deposits where owing to health conditions it becomes necessary to have on the pay-roll about 40% more staff because each employee must have holidays under pay to recuperate.

Future of Dredging.—The dredging industry has to date produced its greatest value from gold. North America has been the most developed and has proved the best, owing to favourable

economic conditions. So far as is known they have reached the peak, but some of the other parts of the world have only fairly started, outside of New Zealand. South America, Africa, and Asia have tremendous areas that are dredgable, but at this date there is not enough reliable data to make it possible to estimate as to the probable amount that will prove workable at a profit. Again, a statement made to-day may prove wholly wrong in the next ten years. Improvements are certain to be made, and some of these should reduce costs, and thus enable lower grade ground to be profitably mined. The investor may feel safe that at the end of the next hundred years the search for dredging areas will still continue.

Safe Investment.—Dredging is among the safest investments to be found. It must be under the direction and advice of men who are experts in that line of work. An engineer who is a good sampler and valuer of alluvials is, as a rule, not the man to design the plant, or, perhaps, to manage it. When sampling has been correctly done the estimate will show to within a few pounds sterling of the gross value. These figures, with the maps and report showing how they were arrived at, should be handed to a competent engineer and a dredge designer. They can work out a close figure as to costs per cubic yard for field and total costs. The same may be said for almost any alluvial mining. Statistics in the United States have been compiled showing that the percentage of all mining failures are about 35% in properly conducted operations as against 95% for all other business.

Standardization.—Standardization is one of the absolute necessities in mining; it not only reduces spare part investment but relieves the worry of sending out parts to the different groups. Experience in Siberian work was that there were five different sleeves or couplings in use on the same group of mines. Some of the distant districts could only receive their spares during the winter season to advantage. During the following summer it was found that those on hand did not fit, and, as it became imperative that the proper ones should be obtained, others were sent out on pack animals 200 miles—at five times the cost of the parts.

DREDGE DESIGN AND EQUIPMENT.

Types of Dredges for Minerals.

Scoop or Shovel.—This was first used, according to records available, by hand in New Zealand. Later a power shovel was operated in the United States and in British Columbia. Only one is believed to have paid its way. This was on the Feather River, California, but it was not a success from an economic

standpoint owing to the low recovery. Small stones would jam in the back and the door or gate would not close, thus allowing the fines and values to leak out. The general design was good. It was mounted on a hull with a hopper on each corner at the bow. The shovel discharged into these alternately on grizzly bars, the over-size going into the paddock and the fines into sluices where they were washed, and the tailing discharged at the stern.

The scoop or shovel design for gold recovery mounted on railway trucks and moved on a track along the face of the ground to be excavated, with the washing plant alongside on a parallel track, was tried out in the Folsom Field, California, about 1900 but was a failure. It did the work, but costs were 10d. (20 cents) due principally to lack of mobility. For open-cut stripping, the design has a use but costs must always be excessive.

For stripping overburden in open-cut work it is practical to use the long-boom power shovel mounted on caterpillar track. The operating cost must be about three times dredge costs under comparable conditions, namely about 1s. (24 cents) per cubic yard, this should not exceed a depth of 30 ft. for a 90 ft. boom. Beyond this costs must increase rapidly. To use any form of dragline excavator under similar conditions the costs may be 20% higher. Each of these machines have their field and when properly installed work satisfactorily but have a lower capacity and higher working costs than bucket dredging under like conditions.

Further reference to shovel practice will be found on pp. 101–105.

Dragline.—This type of machine has been used in a number of places in this class of mining. It was first used about 30 years ago in California by Wm. Destelhorst. The drag scraper worked between two towers, the outer or tail tower being movable over a short swing, and the head tower being at the washing plant. The towers were connected by an overhead cable, which could be raised and lowered from the steam power plant alongside the washing sluices. The scraper was lowered into the paddock or stream and loaded on the out haul (this was poor practice) and then returned by dragging on the bottom and hoisted and dumped into a hopper, which discharged into the sluice. Working costs were 1s. (24 cents) per cubic yard, and the value of the ground was 1s., leaving nothing for the royalties. This work was carried on sometimes in a stream with a tower on the opposite bank, at other times in a paddock.

The Orsk goldfield, on a tributary of the Amur River in Eastern Siberia, some years ago used a dredge of this kind mounted on shore, self-contained, and loading on the in haul. It worked satisfactorily, but costs were about 2s. (50 cents) which left

too small a margin for profit, so it was altered to a bucket dredge and placed on a proper hull.

Machines of the dragline type have been tried in many other parts of the world but rarely have they been a success in metal mining. There are times and places where they are suitable, but, like all methods, cannot succeed everywhere.

Clamshell.—This type will not succeed for metal mining. It can be made to excavate under favourable conditions, but owing to gravel getting in between the parts it will not close tightly, and it cannot be made to dig among boulders where the best values lie. About the year 1900 at Breckenridge, Colorado, a company tried out a plant equipped with one of these on a glacial moraine, but it was a failure; the general design and lay-out were good but it would not handle the material.

Suction Dredge.—The suction dredge generally has proved a failure for mining operations and the type should never be advised for metal mining. Only two successes are recorded one in Michigan, U.S.A., which is recovering from a dressing plant tailing that has been deposited in a lake bed. The material is suitable and is easily loosened by the cutter head. The other is operating in Malay States and is the only success among fifty that were erected.

One of the authors in 1908 was called upon to check this method for the United States Government and found the practice not a success. The general plan was centrifugal pumps mounted on a hull, the work being done in the river. A 6 ft. caisson was lowered to the bed of the Sacramento River and an 8 in. suction pipe extended down to this. A man in a diving suit was lowered inside the caisson with a bar to assist the feed by keeping the material loose and the suction open. Operations were checked for several days and 3 cu. yds. per 24 hours was a liberal estimate for the duty. The diver's statement was that, if material moved, several small boulders would start at the same time and jam in the pipe, and that quite frequently the pump must be stopped to loosen these. His opinion that it would never work was borne out, for all operations were discontinued a short time later.

From experiments carried out some years ago it was found that placer gold placed on a clean surface under water would rise to the suction when it was from 4 in. to 4½ in. above—the pressure being from a hydraulic head of 105 lb. per sq. in.—but if the gold was placed on the ordinary bedrock it sometimes would not rise when the suction was within 1 in., because of a certain amount of adhesion caused by a little clay on the bedrock. Platinum would react about the same but tin and diamonds would be more easily raised.

In general, all minerals are more or less concentrated near or

on bedrock among boulders and clay. There never has been a cutter head designed that will satisfactorily cut or loosen this material, and upon this the suction depends for its feed. As for the cutter breaking up and loosening the bedrock it is most difficult and economically impossible.

Bucket Dredges. -Bucket dredges have been uniformly successful since the beginning. The depth to which any of them

FIG. 5.—MODERN TYPE OF DREDGE. This is capable of working to 75 ft. deep, and is driven electrically, power being conveyed by cable from shore. Steam, steam-electric, or oil-engine electric drives are also used. Close connected buckets have been used on all recent dredges. The tin-dressing system still includes sluices but jigs with distributors have also been employed and this is becoming common practice. As shown the whole is housed with corrugated galvanized sheets. (Supplied by Messrs. Werf Conrad.)

may work is governed by several factors, such as the character of the deposit, value of mineral, etc. For greater depths, greater mechanical strength is required. They have been proved, but it is recommended that, no matter how simple they may seem, the advice of an engineer who has had some experience should be obtained. The more complicated or the greater the possible capital expenditure, the more experience is required. Generally speaking it is best not to assume that any standard design will suit all deposits. Any investor should realize that one of these large dredges is a huge metallurgical plant that will treat and recover the values from about 20,000 tons of material per 24 hours, costing some £150,000 to build, plus freight, and if a plant to treat the same amount from a reef mine were contemplated, the best and most experienced advice would be obtained. No double lift

type should be advised; it only increases the cost of production and does not correspondingly increase the saving.

Dredge Specifications.

The dredge will be (open or close) connected bucket elevator type (state power) driven (spud or line operated) to dig ——— depth below the level of the water in which it floats. It will be designed to operate either on spud or lines. The entire machine to be of wood and steel (or steel). All castings to be of the best material (state kind), special alloys to be used wherever necessary. (The hull to be sectionalized when required for special work and no section to weigh more than ——— tons.)

Fig. 6.—Example of Double Lift Type of Dredge, showing extra long Sluice with Auxiliary Pontoon to Float.

Hull.—The over-all length to be ——— ft. the width ——— ft., depth ——— ft., to draw not more than ——— ft. of water when completed, the machinery to be so placed that the dredge will float upon an even keel, the bucket ladder of a sufficient length to allow digging to a depth of ——— ft. below the water line when extended at an angle of 45°. The whole to be fitted with longitudinal and lateral trusses to give strength to carry the load and rigidity to prevent warping or buckling, with sufficient bulkheads to prevent sinking if the hull should spring a leak. The outer skin to be ——— thick and watertight. The lower deck to be fitted with manholes for inspection. The stern to be designed to take the thrusts of the spuds when in use, allowing for heavy castings and springs to relieve strains. Spacing of frames to be such as to furnish ample strength (the stern to be of the cut-under type when required; this latter may be necessary when there is an excess of fines, where the tailing takes a flat slope. The bow outer corners to be cut back and ladder project a reason-

able length so that the dredge may work into the corners with the least delay and work in a minimum cut when necessary. The whole to be properly ventilated.

Gantries and Trussing.—These to be well designed, and of the best construction for duty required. Bow to have sufficient height and overhang and tied to main centre chords, and to have 4 solid foot connections with the deck of a size and strength to carry the load. The main to be a part of the two centre chords and strutted crosswise, and of a strength to carry the digging ladder and driving gear for the bucket line, main hopper and its load, also the upper end of the screen, etc. The stern to be of a strength and height to carry the spuds, sluices, and stacker and to be well tied to the centre chords.

Buckets.—These to be of a ——— cu. ft. capacity (close or open connected); there will be ——— eyes in front and ——— at theback. The front eyes to be slotted on the outside to engage the lug on the pin, the back eye to be slotted to take a ½ round bushing. The back and hood to be cast in one piece with shoulde ron the outside of hood to engage the lip, the inside of bucket to be without corners. The whole to be of a special alloy steel suitable for heavy work.

Lip.—Attached to the hood by heavy rivets a special alloy steel lip of great strength and wear-resisting qualities, the back of the bucket to be designed for round or six-sided lower tumbler. Inside-outside type is best.

Pins and Bushings.—First of a special alloy with lug. Second of a suitable wear-resisting alloy.

Lower Tumbler and Shaft.—(The best practice is a round tumbler.) The tumbler to be of a special alloy steel and body to be shrunk or pressed on the shaft. Hollow shaft to be of a special steel and size for ample strength.

Upper Tumbler.—To be six-sided of good alloy steel shrunk or pressed upon a special steel shaft of size and strength for the work in hand, the tumbler body to be cast and recessed for wearing plates (these to be of hard tough steel), and face of tumbler to be machined.

Digging Ladder.—To be of a suitable length and of the solid plate girder type, bulkheaded and stiffened where necessary, with a spill plate full length on top to catch and convey back to the lower end all spill from the buckets. Lower end to be fitted with two extra heavy bearings to carry the lower tumbler and designed to prevent spreading. The upper end to be fitted with extra heavy bearings to carry the ladder. The sides of the ladder to have suitable chafing beams to work against chafing beams alongside of the well-hole in the hull (an idler to carry any extra long bucket line).

Ladder Rollers and Bearings.—On the top chord of the ladder these are to be placed in suitable adjustable bearings of good size and strength. The rollers to be not less than ——— inches in diameter and — ––– inches long, of a high-grade alloy steel shrunk or pressed on suitable alloy steel shafts. (The adjustable bearings are to enable the bucket line to be kept in alignment.)

Ladder Suspension.—On the bow gantry-cap will be carried suitable sheaves in a series of yoke bars to take the cables to correspond to sheaves and yokes on the lower end of the ladder. All to be of great strength to support the total load. The lower end of ladder to be carried by 2 sets of suspension members.

FIG. 7.—EXAMPLE OF AN EARLY TYPE OF BOW GANTRY AND DIGGING LADDER HANGERS, SHOWING ROBUSTNESS.

Main Drive.—This to be of the double gear type, of suitable size and strength of the best cut steel gear and the shafts to be of special alloy steel of a suitable size and strength. (If dredge is for heavy duty then the main gears to be of the rim and spider type.)

Main Hopper.—To be built under the main drive to take the material as dumped from the buckets and of suitable size and strength; the bottom of hopper to be lined with suitable alloy steel bars to withstand the wear.

Save-all Bars.—Just under the main hopper and upper end of bucket line these are to be placed so that any boulders spilling over will be directed into the pond and the fines will go into the save-all sluice.

Revolving Screen or Trommel.—This will be built up in sections, so designed that any perforated plate may be removed and another inserted. All such plates must be interchangeable. The total length of screen ——— ft., the effective screening length ——— ft., the size of holes ——— in. (size of holes are governed by the size of particles to be screened and the quantity of fines). The end shells to be joined by suitable bars to carry the load. Inside to be fitted with circular retarding rings and longitudinal bars to prevent the gravel from sliding inside the screen. Outside will be fitted with two circular tread bands to carry the screen and its load.

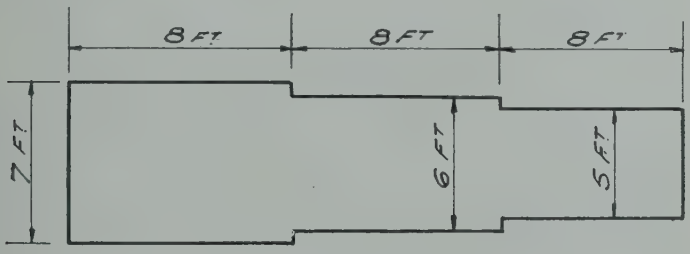

FIG. 8.—STEP TROMMEL.

Screen Drive.—To be of the usual heavy type, the main drive shaft direct-connected to the motor. Bearings, shafts, and rollers to be well made and of good material, the whole carried in a strong heavy structural steel frame.

Distributor.—Around the screen to be built a housing and suitable launders to distribute all material equally to the sluices by arrangement of suitable gates. Wearing plates to be arranged to be replaced at points of excessive wear.

Revolving Screen Hopper and Over-size Chute.—From the lower end of screen a hopper will be placed so that it will direct material into the chute that discharges on to the stacker, which chute is equipped with riffles when necessary. All to be of a design that will allow of wearing parts to be easily replaced and strength to withstand rough usage.

Tailings Conveyor or Stacker.—This to be of a length and strength for duty required, provided with suitable troughing idlers to carry the belt in use and support the motor for the drive at the outer end, this belt to be of a standard make, the whole

to be attached at the stern of the hull and suspended from the rear gantry and to be of the lattice girder design.

Winch.—For manœuvring the dredge, digging ladder, spuds, etc., ——— drums and located at a convenient point near the bow on the lower deck, and if need be with bars and levers so that it may be operated from the pilot house and with suitable clutches, brakes, etc., for a one-man operation. To be built extra strong and of good material (usually drums are wanted for 4 side lines), 1 head line, 2 spud lines, and if a small dredge 1 digging line, but if larger size then the head line and ladder lines should be on a separate winch. Number of speeds to be specified.

Spuds.—These to be of a suitable size and length (of steel where possible), and to be equipped with a cast steel shoe. All rivets to be counter-sunk on the outside, thus exposing a smooth surface, and to be fitted at the top with suitable yoke and sheaves of ample size and strength; each should be of the same size.

Headline.—To be equipped with ——— ft. ——— diameter, flexible steel rope.

Cranes and Travellers.—All cranes and derricks needed to be supplied on the larger dredges with an overhead track to overhang the bow so that spares may be picked up on shore and carried to points on the dredge where needed and lowered into positions. Suitable hatches must be arranged over the main hopper and screen (see Fig. 9 for track arrangement).

Hydraulic Press.—(For use on larger dredges). To be fitted so that it may grip the bucket line and be used to push in and out pins and bushings.

Fairleads and Deck Sheaves. These for the proper guiding of lines, etc., must be correctly designed and of large diameter in order to minimize the bending of the ropes, and with good strong bearings to withstand the wear, and to have suitable safety protectors.

Ladders or Stairs.—These not to be too steeply inclined and all metal treads must be checkered to prevent danger from slippery surfaces. There should also be a sufficient number so that too much time lost in running about looking for stairs may be prevented.

Footbridge.—To be supplied of a given length and width and supported from the bow gantry, and raised or lowered by a hand or power winch.

Platforms.—A sufficient number of these should be provided wherever required (large dredges require more than small ones).

Housing.—A suitable superstructure to support all machinery and the housing must be provided for.

Pumps and Piping.—Three centrifugal type, one low-pressure, one high-pressure, one for fire purposes. Low-pressure for the sluices, high for screen and the fire pump as an auxiliary. Low pressure about equivalent to 30 ft. head, the high a 60 ft. head, and the fire pump a 90 ft. head. The piping to be arranged to supply sluices, the spray pipe in screen, hopper, and jets to buckets.

Wire Ropes and Belting.—All running ropes for winches, etc., to be high-grade flexible steel of a suitable size and strength, all standing lines of a high tensile strength but not flexible steel. Belting of a size and quality for duty required.

FIG. 9.—EXAMPLE OF OVERHEAD TRACK. The dredge illustrated is Canadian Klondyke No. 3.

Electrical Equipment.—The entire dredge to be operated electrically (if possible), 3 phase, ——— cycle, ——— volt apparatus. Certain small motors to be ——— volt, with ——— volt for lighting. It is preferable to have the small motors the same voltage as lighting to reduce the number of transformers; sufficient of the latter to be supplied.

Motors.—The main drive to be slip-ring type ——— h.p. ——— r.p.m. reversible, same torque at any load, to operate from one-half to full speed.

Winch Motor.—To be ——— h.p. ——— r.p.m. slip-ring type, reversible, to operate from one-half to full speed.

Screen Motor.—This to be ——— h.p. ——— r.p.m. slip-ring type, reversible (so that direction of rotation of screen may be changed in order to give a more regular distribution of feed to the sluices).

High-pressure Pump.—To be ——— h.p. ——— r.p.m., squirrel-cage type, complete with starting compensators and over-load relays.

Low-pressure Pump.——— h.p. ——— r.p.m. with same specifications as high-pressure.

Auxiliary or Fire Pump. ——— h.p. ——— r.p.m. ——— volts, same specifications as high-pressure.

Transformers.—Number ——— k.w., ——— volts, ——— phase, ——— cycle. Complete with fuse, cut-outs, etc.

Switch Board.—Panels to control all motors, lighting, etc.

Supplies.—Under this head include all panels,——— ammeters, transformers, volt-meters, disconnecting switches, for incoming lines.

Cables.—Wiring and cables which may be necessary for all power and control of same, with shore cable, all to be installed in a thorough fireproof and workmanlike manner.

Accessories.—Switches, sockets, lamps, and all other electrical fittings ample for a dredge of this size, and signal gong.

General.—The dredge to be built with a view to making it a most complete and substantial machine of its kind.

Heating Plant.—(If in a cold country.) The steam boiler to be placed in a compartment of the hull, if possible, with a seacock for flooding if a fire should occur. The boiler to be connected with piping, all parts to be heated to have ample radiators connected to piping, with piping leading to the stacker (so it may be heated) and to the deck, where steam may be required for cutting ice from the deck and bucket line, also for warming the sluices if needed.

Bucket Dredge Designs.—In regard to design, the sluice-box type is the least complicated and lowest in first cost. It may have either an open or close connected bucket line. The simplest form is mounted on a hull floating in a paddock, built without a screen. The buckets dump on to a grizzly that discharges the oversize at the side, while the fines pass into a sluice that discharges at the stern. The lift must be high enough to allow for grade and dump. When conditions are right these dredges work well, but if more boulders and some clay are found then it is better

practice to place a trommel for sizing and washing. It is not, however, advisable to have a shaking screen on any dredge.

The standard dredge design may be either an open or closed line, depending on duty required. Experience advises either type according to the character of the deposit. Open lines as a rule work best among boulders. A closed line may be made to work better in such ground by a slight change in the bucket design, cutting the hood side of the bucket back slightly (Fig. 10)

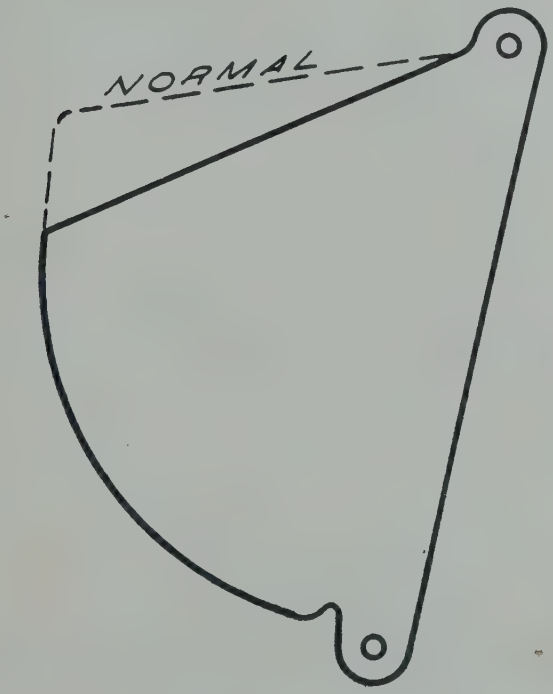

FIG. 10.—BUCKET CUT BACK ON HOOD.

so that the boulders when picked up will not jam between the lip of one bucket and the hood of the other, sometimes causing breakage. The closed lines as a rule have from 40 to 50% greater capacity for the same sized bucket.

Each part should be carefully checked over by experienced designers and builders, as it is good practice to have the best of everything, it being seldom advisable to use poor designs or material. Transport and erection costs are the same per ton,

while from the operating point of view there can hardly be a comparison. Losses on renewals and replacements during what should be producing time is the one thing to be avoided. The designs of all parts should not be too complicated, or, if they are, the technical ability of the dredge crew must be higher, thus increasing the operating cost.

Boulders picked up by the buckets may be broken by dynamite exploded on the boulder immediately after it clears the surface of the water. The amount of such explosive required will soon be found out in practice. This method of breaking will do little, if any, damage to the bucket line ; alternatively the large boulders may be raised just above the deck level and then rolled on to the deck and when the dredge swing reaches the corner of the paddock they are dumped overboard into the corner where they give little trouble afterwards.

The dredge may be so designed with grizzly bars and a side chute at the top of the main hopper to remove the boulders but such are not to be considered advisable.

Boulders too large to be brought up by the buckets, may if not too numerous, be crowded into the corners, or if too big to be moved, may be buried by digging alongside of them and allowing them to roll or slide into the hole. Boulders 6 ft. to 8 ft. in diameter have been disposed of in this way. The richness of the deposit and the character of the bedrock must decide how far this extra expense is justified and may limit such alternatives.

Shore Lines and their Uses.—The steel ropes used as shore lines should be of the best material and suitable for such duty as may be required of them. The head lines should always be of sufficient length to enable the dredge to be manœuvred without cramping the operator's efforts in any way. If the dredge is in an open stream of large size, then two head lines must be used and they should be carried on floats of some sort. It is not good practice to attempt dragging them along the bottom, for they may meet obstructions and hang up, causing vexatious delays. If on floats they will require some attention to prevent floating objects being caught.

If the dredge is operating in a paddock the line should have some sort of roller to carry it over obstructions. Head lines are not required if working in small valleys, but are an insurance against damage if the valleys are subject to sudden freshets. Instances are on record of several dredges being wrecked when working under such conditions without these lines. They may be used to work against instead of spuds.

All dredges should be equipped with two bow and two stern

lines, which should be of such length as to fulfil the requirements of the work to be done, that is, long for wide cuts. In operating, they should be crossed (see Fig. 11), first because it enables the dredge to move forward or backward, and if digging on the headline to prevent the bucket line from being rammed into the bank, thereby reducing strains and wear by the surging on the head lines; second, if digging on the spud it is easier to step ahead or over for the next cuts, with a minimum of lost time. The old way, by the use of spuds for stepping ahead, consumed at least three hundred per cent. more time.

Washing and Recovery Plant on a Dredge.—The area of the washing and recovery plant on the dredge must always be ample for the work in hand. For gold recovery under average conditions

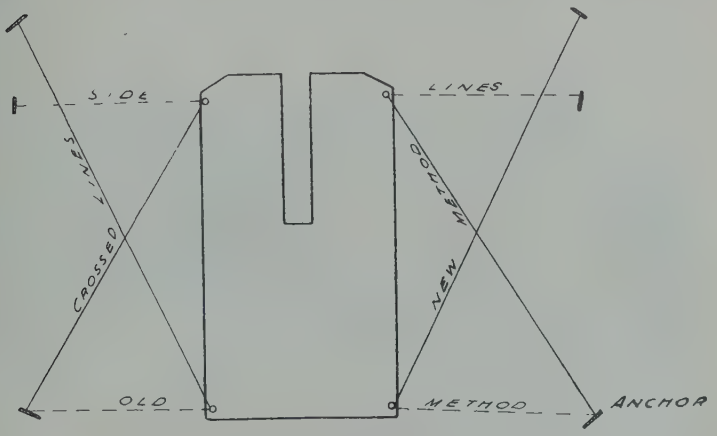

Fig. 11.—Crossed Lines.

about 16 sq. ft. of sluice per cubic yard per hour is good practice. For platinum the area would be the same, but for tin more should be allowed, since practice in Portugal, Africa and Malaya has shown that jigs may be used to good advantage and reduce the table or sluice area to a minimum. This can be applied to other dredging. The practice of using jigs is steadily increasing.

To obtain the highest efficiency the feed of fines and the water supply must be regular and the table gradient must be suitable. If these are correct the particular design of tables is not so important. On gold dredges, where the tables discharge at the stern and the hull has sufficient free board, it is an easy matter to regulate the grade of the tables by the use of ballast. The same holds good for tin.

It has been proved for tin dredging (such as in Nigeria where the overburden carries no values) that it is better practice to have the sluices made of the truss type, the centre of such truss hinged at the stern, with a quick-acting adjustment at the forward end and a stop above and below. Thus, when the dredge has been stepped up and is ready to begin a new slice into the barren overburden, the winchman can signal by means of the gong to the lower-deck men who at once give each lot of sluices the maximum gradient, in order that the material may be rushed through, thereby increasing the digging capacity. On reaching the wash containing the values the operation is reversed by use of the stops above and below. There can be no mistake as to too much or

Fig. 12.—Example of Trussed Sluices which are easily adjustable for Grade (Ropp Mine).

too little shifting, as each truss carried all the tables on one side of the dredge. Such arrangements could be made on other classes of dredging.

The catch or save-all sluice must be arranged so that it will take care of all the spill from the buckets after dumping and possess sufficient grade to discharge at the stern.

The number of " clean-ups " for each class of mineral must be worked out during operations, as conditions are not always the same and no fixed rule can be made.

In gold-saving, mercury will be a great help at times; in other deposits, of very little use in the sluices. After the heavier, cleaner gold has been recovered, then the fine gold and black

sands should be put into a tube-mill and ground for a time, the length of which must be found by experience. The grinding is for the purpose of brightening the surface of the gold so that when the mercury is placed in the charge it will more quickly amalgamate. The addition of certain chemicals will assist this, and the best for any particular case must be found by practice. Some operators prefer cyanide, and good results have been had by the addition of wood ashes.

For gold saving there are many types of riffles used and a great many opinions exist as to the merits of each. The opinion is given (and this is based on practical experience) that the type of riffle is not so important as the careful adjustment of feed and water to the sluices for gold and platinum owing to their great specific gravity. The riffles should be designed so that they may be put in in sections to expedite handling during clean-ups.

For saving tin wider sluices with a 2 in. by 4 in. riffle on edge across the flow about 7 ft. apart have been tried with good results (if no jigs are used). A mechanical agitator moves up and down the sluice, the bar clearing the bottom by about $\frac{1}{4}$ in. and having a travel of some 18 in. The bar should have a diameter of at least $\frac{1}{2}$ in. This keeps the fines riffling as the bar moves forward and back. These agitator bars must be made adjustable, so that the best point for placing them may be worked out in practice. Several types have been tried ; any of them will as a rule be more satisfactory than the human element, for a mechanical device works all the time and requires no time off. Clean-ups for tin should be almost continuous, the richness of material worked controlling this. A chute should be arranged about half-way down the sluice, closed easily by a trap door. The chute empties into a launder that carries back towards the bow of the dredge and discharges into an elevator boot. The elevator raises the rough concentrate to the jigs where it is reduced in bulk, and this final concentrate is taken to the dressing plant on shore.

Several attempts have been made in Brazil to mine diamonds by dredging, but none has proved a financial success. The authors' experience in this class of mining is that owing to their low specific gravity they will require a most complicated washing plant of trommels, jigs, etc., so that from an economic point it is not a good investment unless the ground is fairly high grade, say a $\frac{1}{4}$ carat per cubic yard of average stone (8 stones to the carat).

Generally the amount of mercury to be used in gold recovery must be worked out in practice. So also must sluice areas, etc., because as the character of deposits varies so does the amount of fines decrease or increase, and sluice areas must be arranged accordingly. A few simple tests should be made so that an

approximation as to the percentage of fines may be known, also the size of the particles of mineral to be recovered. An experienced engineer will be able to estimate these things in a very short time, so that screen perforations may be correctly dimensioned.

Cold and ice have an important effect on recovery. It has been proved by tests that particles of gold are among the first to accumulate ice in the sluice, thus causing losses among the flaky

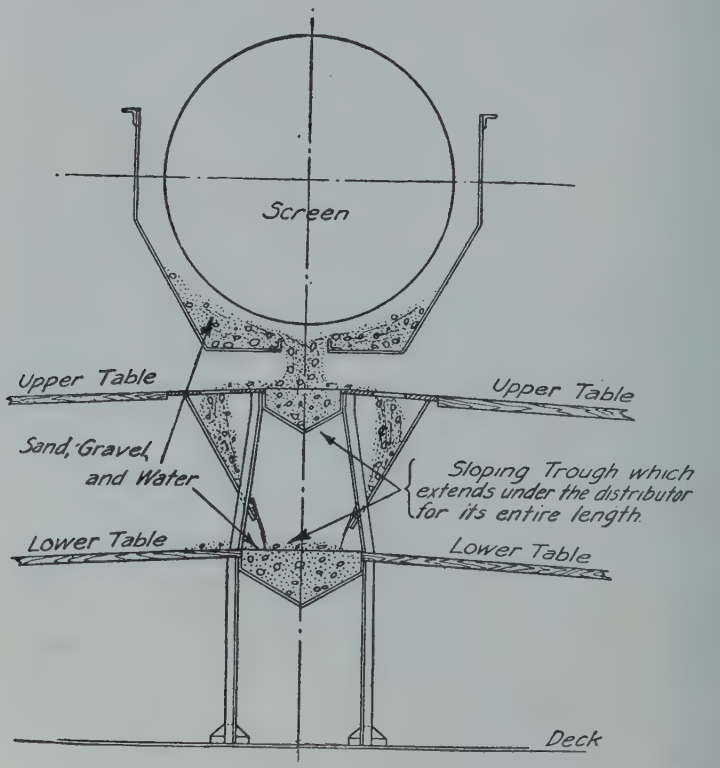

FIG. 13.—SECTION OF DOUBLE BANK SLUICE AND TROMMEL.

pieces. Close attention should be given to the temperature of the water in cold weather to prevent this.

Poor recovery of values may be due to sluice troubles and to hard rough bedrock or an excess of sticky clay. If the recovery for gold or platinum averages 5% below the sampling, and this sampling has been properly done, then a close inspection and analysis should be made. For tin 10% may be allowable.

On certain dredging work in Oregon, U.S.A., some years ago (for gold) the recovery was near 105%; the factor used in sampling was a bit too conservative. At Dawson in the Klondyke the dredge recovered a fraction of 1% below the values disclosed in sampling.

One tin dredge in Nigeria recovered about 40% above the sampling; this, too, may be taken as being due to ultra conservative sampling. Such conservatism would have condemned a low-grade property, but great care and good judgment would prevent such results being obtained. A tin dredge operating in Portugal for over 10 years has recovered more than the sample average, because of the lack of practical experience in tin dredging, too conservative a factor being used in the final estimate.

Double-bank sluices may be used where it becomes necessary to increase the sluice area beyond the ordinary, but as a rule this applies only to gold or platinum. Ample water must be arranged for—approximately 1,200 imperial gallons per cubic yard washed per 24 hours, though the amount will vary in the same deposit. This does not mean that that amount of fresh water is to be added to the supply. The fresh water to be added may be taken at 50 gallons per minute for a 3 ft. dredge and for larger sizes more. Clayey ground will require more than clean sand and gravel, and an excess of fines, too, increases the demand for water. At times, it becomes necessary in working loose porous ground to increase the water supply so that the flotation of the dredge may be maintained at a proper elevation.

One of the authors has experimented with undercurrents on hydraulic sluices, on hydraulic elevator plants, and on dredges and has checked them up on other operations, but never have they been found returning a reasonable profit. If the main sluices and riffles are properly laid out and designed and the feed with water supply well regulated there is seldom need for this addition to the plant.

Plan and Elevation of Dredge.—Fig. 14 gives a general idea of the plan of setting the machinery and also the distributor and sluice. The stacker is left out in the plan but it may be seen in the elevation. The side view gives an idea of how the drag of the buckets on the bedrock may scrape and clean it. Working on the swinging cut each bucket removes a part as it passes, this being crowded into the receptacle. During the latter operation the swinging speed is reduced so that a better recovery may be made. This demonstrates the finish of one slice down the face. The ladder is then raised above the bank and a step ahead is taken for the next swing.

In this elevation the bow gantry is shewn as being connected to the top truss by wire rope but it is better practice to make this tie with a solid beam. The spud is indicated in the tailing,

which is correct and best, because the spud is not lowered so far, and if any unusual surge comes against it this is partly absorbed by the spring in the spud and partly by the give in the tailing.

It is good practice, when making a turn and lowering the spud into deep water with the shoe resting on bedrock, to dig carefully until there is a heap of tailing around this as a protection from heavy back-surging against the spud. Cases have been known of heavy steel spud being badly bent because this precaution had not been taken, and to remove, straighten, and replace it, caused a loss of almost half a month's production. A very good general scheme for the support of the stacker is given.

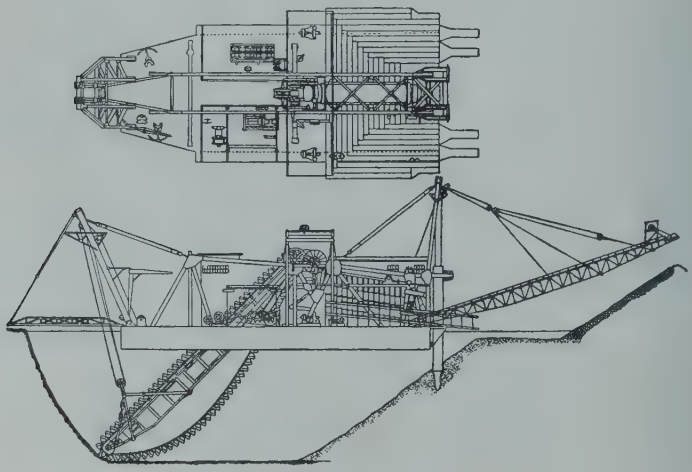

FIG. 14.—PLAN AND ELEVATION OF DREDGE.

Dredge Erection.—In this class of work, when possible, the hull should be assembled adjacent to the paddock where it is to be floated. Launching from poorly constructed slip-ways is difficult and expensive. Thus, instead of a few minutes, as is the usual estimate, hours have been waisted at times owing to some unforeseen weakness.

Short cuts may be worked out, too, that will reduce this cost. During 1912 the Canadian Klondyke Company at Dawson were erecting two 17 cu. ft. dredges in the Klondyke Valley at the same point. One large paddock was excavated to hold both, with a timber-cribbed double track laid through the centre on which two locomotive cranes operated, heavy pieces being picked up in the yard and brought to the erectors, thereby saving a lot of

time and £4,000 ($20,000) on the total cost. These were dredges with timber hulls. The man-hours for such erection were 150,000 each. By this is meant that one man working could build a dredge in that number of hours.

Size of Dredges.—The size of dredges should always conform to the duty required; if the conditions are right, a one cubic yard bucket is advisable. The operating costs per cubic yard are reduced on the large dredges since the same labour is capable of handling the bigger yardage. The wear varies but little, and the power consumption is about the same per cubic yard. The depth to which a dredge may dig below water line is controlled by conditions. These being favourable it is possible to work 150 ft. depths, and the requirements are mechanical in so far as the dredge is concerned.

Capacity of Dredges.—It is an easy matter to compute the capacity of dredges theoretically. The size of bucket multiplied by the number per minute, reduced by 33% so that the calculation may more nearly represent the material in place in bank. Actual high duty is not impossible but most difficult, owing to the inability to obtain full buckets, and variation of operating conditions. This will be from 80% to 90% of the possible running time on a modern, well-constructed dredge. There are also the personal equation, night shift, etc., to be considered. Generally there are 3 winchmen per 24 hours. One man may average well filled buckets, another about 75%. Generally, but not always, the night shift has the lowest average.

The washing capacity must always be considered also, and the plant must never be over-loaded, in order to have efficient saving of values. On encountering a high-grade pay streak, it is always advisable to slow down on the digging.

Some years ago on the Ropp Mine, Nigeria, such consistently poor yardage was made, and the percentage of recovery was so low that, after due consideration, the night shift was discontinued. The results were much better after this. It is a well-known fact that the labour efficiency in the tropics is not of the best, but there have been operations in temperate zones where such tactics would have, without doubt, improved the recovery.

Life of Hulls.—The life of a hull must vary according to the material and timber used, and its treatment, also whether in temperate, tropical or northern zones. In early-day construction, timber was used and by practice it was found that a good Douglas fir skin would last nine to twelve years in the temperate zone, and it has been determined that by treatment under the same conditions the hulls would last eighteen years. It is not advisable to use timber in the tropics, except when treated to withstand rot, and even then it is not good practice.

In the Klondyke some twenty years ago a dredge hull was built of Douglas fir that had been logged in the salt water near Vancouver, B.C., and had become quite well saturated before being cut into lumber. The dredge after working for about seven years was dismantled and moved to another area, and it was found that most of the skin material was damaged in dismantling too much for re-use, although all the superstructure and larger timbers were in a good state of preservation, and holes bored in the large pieces for tests gave no signs of decay. Some of this preservation may be taken as due to the pickling in salt water and some to the fact that all material in the North is in " cold storage " for about seven months each year.

It is the best practice to use steel—other conditions being equal. Generally all hulls must be well ventilated in order to obtain a maximum life for them and the superstructure and all parts above the water should be kept well painted. Attention must be given to the repair and maintenance of hulls as much as to any machinery. Paint is one important item, and keeping bolts and rivets tight is another. The salvage value of the hull is much greater if it has been well looked after.

It should be generally realized that hulls must be kept on an even keel (frequently they are not) as no sluice will make an efficient saving with the feed running on one side. With machinery properly placed it is a simple matter to place a small amount of ballast to correct this error. On tin dredges balance tanks have been provided to correct movements due to the accumulation of material in the sluices.

Sectionalized Hulls.—Sectionalized hulls are to be considered when small areas of limited yardage that are better suited for this class of mining are to be worked. The working will be less expensive per cubic yard than gravel pumping or dry land excavators. The latter two will cost less for installation, but the time lost due to many moves, when no production can be had, makes everything in favour of the dredge. Also fewer men are wanted and less horse power is consumed on the dredge.

There are several ways in which these sections may be bolted together. Some design must be used that is not too complicated but all this as a rule can be left to the dredge builders. It is only necessary to state that no section may weigh above ——— tons (that weight being about the maximum that could be transported from one area to the next), also stipulating that all sections are to be bolted to each other—not riveted—and that each section is to be fitted with seacocks below the water line so that water may be allowed to enter in order to trim the hull more easily.

Care, too, must be taken in all dredges that the hulls are large enough to furnish plenty of freeboard. This mistake

occurred a number of times in the early days of designing but seldom does now. If it should, one or more small pontoons may be built and attached to the main hull, but as a rule this makes a poor job and an unwieldy dredge. The modern dredge designer is remarkably well informed on these practical points.

Spray Pipes and Nozzles.—On average ground preference is given to the spray pipe for washing inside the screen. The pipe must

FIG. 15.—BACK OF BUCKET FOR ROUND TUMBLER, SHOWING ALSO RIBBED BACK EYE.

be so designed that the holes are the right size and arranged that it may be easily cleaned of grass, roots, etc., which may tend to obstruct the spray. The adjustment of the pipe must be such that all the spray will reach the material at the point to do the greatest good. If the material is of a sticky nature, then it may be advisable to have both spray and nozzles. The efficient working of either depends upon a good water supply and pressure; also on preventing too much obstruction from debris.

Bucket Bushings.—Some operators prefer the round bushing. These must be designed with a lug to prevent them rotating inside the back eye. The half round bushing is preferred by many because it is more easily ground for fit and smoothness. They may be punched out and reversed so that more wear may be had, for by so doing more life per pound of metal is obtained. The round bushing has twice the weight of the half round bushing plus the lug.

Step Cylindrical Screen.—(Fig. 8, p. 119.) Some of the advantages of the step cylindrical screen are that a reduction of weight may be had of about 20%, the drive may be at the head, and the main feature is that in material containing clay the washing is better as each 6 in. step retards the travel of the feed, and there is less wear on the plates than in the straight screen where the retarding ring segments are used with their attendant drop. There are more fines discharged on the upper part of the distributor by the step screen, and the design of the sluice and screen holes must be changed slightly to take care of this excess. More care must be given to the design of the step screen where strength is required for heavy duty. A greater number of spare plates must be carried, for the sets are not interchangeable. Generally the straight cylindrical screen is the best for all purposes, as retarding rings and parallel bars may be taken out or reduced in size when necessary.

Small Winch and Motor on Dredge.—An independent rig of this kind is advisable where timber is to be cleared from the front of the dredge and sometimes from the paddock. Also in the far northern countries it may become necessary to remove ice. A light line may be used to shift the head line and side lines. With a line of proper size and snatch blocks, many bits of work may be done on shore that would otherwise be done by hand.

This winch is much better placed on the dredge than on the bank, as it is always up to the work and if on an electrically operated dredge is there close to the transformer.

Power.—In making estimates for power supply several things must be taken into consideration, such as the extent of the operations, life of the property, etc. Steam as a rule comes first in out-of-the-way places, that is, if fuel may be had at a reasonable cost. The flexibility and simplicity of this class of power is all in its favour, but no dredge should ever be constructed with a single engine, because there must be a variation of speed due to digging. This will cause the pumps to deliver an intermittent supply of water which affects the efficiency of the sluices, causing losses, more in tin than in gold. It is best, provided the business will warrant it, to use one engine and generate electricity on board, driving each machine by direct connected motors.

Internal combustion engines may be advisable at times but, if conditions justify the outlay, electric generation on shore in a central station with distribution to the dredge or dredges will simplify operation on board. Hydro-electric power is the ideal, and should be considered if water is available. The initial capital outlay is high but once installed, power costs are at a minimum. Electric power per cubic yard dredged runs from $\frac{3}{4}$ to 2 k.w. Hardness of the ground, tough clay, etc., will regulate this, and an experienced dredge engineer can arrive at a close estimate. Small and large dredges seem to require about the same kilowatts per cubic yard dug.

Fuel Supply for Power and Domestic Purposes.—In considering fuel supply it is well to remember that fuel must be provided to cover the probable life of the property. Instances are known of operations beginning with steam power and calculations being incorrectly made so that later the native timber was found not to meet the demand, partly owing to more fuel being required (to generate a given horse power) than was allowed for in the estimate, and also to other work in the immediate vicinity demanding fuel. Many tropical woods are most unsuitable for fuel purposes.

Domestic supply must be obtained for every possible purpose. In the tropics where fuel is only wanted for cooking the problem becomes much more simple, but in the Arctic regions it is a very serious problem, for as a rule the coal supply is of poor quality and the cold seasons are long. Where it is necessary to heat all quarters for warmth, greater allowances must be made. In Siberia, for instance, large stone or brick stoves are built as part of the house and one-fourth of a cord of wood is put into this stove at one time, this being done several times a day to get the stove warm and a large heap of live coals ready for the night. After this the draughts are closed so that the stove will retain the heat until the next day, for seldom does a winter pass in the Lena Goldfields region when the temperature does not drop to 70° or 80° below zero Fahrenheit. These mines have been worked for over eighty years almost entirely by drifting, and large amounts of timber have been used. Great areas of forest have been cut down and about 25% of the mining costs were for such timber. At the present time dredging is beginning and the fuel is being brought from 150 miles away, partly by floating down a river and partly by rail. Somewhat similar conditions are found in the Klondyke and Nome country.

DREDGING PRACTICE.

Factors Influencing Dredging.

Gradient.—Deposits have been dug that had a 9% grade. The depth governs this, for, if too shallow, an excessive amount of bedrock must be dug to allow for flotation or dams constructed to permit the working of the dredge. 1 to 2% is much the best for this class of mining, 9% making a fine gradient for hydraulicking.

Hardness of Ground Dredgable.—Material has been worked in the Folsom field of California of a gravel scoria nature that was hard picking ground. The dredges were of a heavy design, as required for such work, and they could dig the bank, but by the use of boreholes and dynamite the ground was shattered at a cost of about 1d. (2 cents) per cubic yard, thereby greatly increasing the capacity and reducing the wear on machinery and demonstrating that it is economically a good practice to blast tight ground.

Here, too, it was found advisable to place a hydraulic monitor on the bow of the dredge, supplied with water from a high-pressure centrifugal pump to keep the banks above water-level cut down and prevent them caving and falling upon the digging ladder and blocking the bucket line. The effective range of the water jet is about half the head. It is good generally to have a monitor on the bow so arranged with an independent motor so that it can be used to cut any bank that may stand at a dangerous height above the water. Once the bucket line is caught under a caved mass it is a most difficult matter to get it free, and there is also the danger of broken parts and a bent or sprung ladder.

Whether to dig to bedrock must be determined by practice, but in some cases is unnecessary. Generally, where the conditions have been determined by an experienced field man, there should be no question of doubt after a careful analysis of the report. The best practice, of course, is to dig to bedrock.

A short while ago the Yuba Consolidated of California, dredging in the Yuba River Valley, was visited and it was found in one part of the field that they were digging to a clay sand streak about 35 ft. down and were making a profit. To dig to bedrock would have entailed a loss.

In order to work a false bedrock successfully, there should be a clay or semi-cemented streak. To do this, the dredge must be most carefully handled and it is generally advisable to operate with a spud, because the dredge is stepped up for the cut with the bucket line. Extend above the bank, then lower the bucket line a foot or more on the swing, then begin digging. On reaching the point for return the line is again lowered and digs on the

return. When the false bedrock is reached the lowering of the bucket line is less and digging speed is consequently reduced. It is most difficult to attempt this method on head line work.

Digging Bedrock and Cleaning.—The question will be as to whether the dredge will be able to dig bedrock and clean and recover the values. A modern well-built dredge of the best materials will dig any bedrock that can be taken up with a sledge and gad, and the bucket lips will thoroughly scrape up all material that is worth while from an economic point of view.

About 1911 the Canadian Klondyke and the Yukon Gold Companies carried out some careful tests to determine if the dredges would take up the bottom and how deep it would pay to go into the bedrock, most of the formation being a rather hard silicified blocky schist with occasional narrow igneous intrusions. The first-named Company found, after digging to a depth of 5 ft., that while an appreciable amount of gold was recovered, the wear on machinery and reduced capacity did not warrant this, so they reduced their average digging into bedrock to about 1 ft., while the Yukon Gold Company, after going to a depth of 12 ft., later reduced their average to 5 ft.

The Three Friends Dredging Company, at Nome, Alaska, about 1910, working a blocky siliceous limestone, found they recovered two shillings (50 cents) gold per cubic yard, to a depth of 12 ft., but there was no profit in working to that depth, so they dug an average of 5 ft.

Experiments were tried by the Canadian Klondyke in 1911 and 1912 to determine how well the bedrock was cleaned. This work was carried out by means of a diver, wearing a regular diving suit, who was sent down regularly to take samples from the top of bedrock behind the bucket line. This was continued for one season on one $7\frac{1}{2}$ cu. ft., and three 17 cu. ft. dredges, and the work proved that no values worth while were being left.

Spud and Headline Operating Compared.—This is a question on which there is quite a diversity of opinion and much has been written on the subject. It is the best practice—where possible—to have the dredge equipped with both head lines and spuds. Under conditions where a steady cut on the swing is required, it can best be obtained by the use of the spud. On the other hand, by the intelligent use of all the lines it is possible to attain good results without the use of spuds. Generally speaking, however, the spuds seem to be more in favour among dredge operators.

Neither method is the best under all conditions. If there is a stiff "puggy" clay or soft overburden such as on the tinfields of

Northern Nigeria, work should be done on the spud, because by lowering the ladder only a few inches and swinging slowly in order to cut the material thin, it then rolls up in the bucket and when this dumps its contents the sticky clayey material comes away more cleanly ; and, if in working alluvial ground containing fine and flaky gold, by the use of the spud it is possible to hold the dredge steadier and prevent the surging that occurs on the headline while maintaining a more even flow of water in the sluices, which tends to give a higher percentage recovery.

Lost Time.—The lost time and delays as considered under this heading are those that occur when mineral-bearing material is not being raised.

	Percentage.
Stepping-up	13·6
Stacker	4·2
Clean-up	11·7
Motors	1·0
Lower tumbler	8·2
Upper tumbler	4·4
Lines	5·1
Pumps	3·1
Screen	7·8
Buckets	8·1
Power	6·6
Rollers	4·6
Sand	0·2
Winches	2·0
Bucket drive	2·9
Miscellaneous	16·5

Stepping-up should include anything that has to do with shifting the dredge into a new position.

Stacker should include any delay of discharge of material by these parts.

Clean-up includes any part of this work.

Motors include all delays brought about by any of these.

Lower tumbler includes any adjustments to this part of the structure, oiling, etc.

Upper tumbler includes oiling, adjustments and replacement of wearing plates.

Lines relate to all lines on board or ashore that are in any way connected with the dredging operations.

Pumps relate to water supply in any manner.

Screen delays concern anything to do with sizing the material.

Buckets concern repairs, replacements, pin renewals, etc.

Power relates to any delays due to lack of power, breakdown, or renewal of parts.

Rollers. The loss of time under this head may be distributed with the bucket-line and stacker.

Sand. There will always be a question here. If an excess of

this occurs, it is often readily overcome by a slight adjustment of the pump without loss of time.

Winches. Any loss of time occasioned here is charged against this item.

Bucket Drive includes loss of time due to main gears and belt bearings.

Miscellaneous should cover items not included in the above.

The returns given above should be gone over carefully each month and a complete analysis made from the records in the log-book. By so doing, better material might be used or slight alterations made that will reduce the total lost time. Lack of sufficient suitable spares is the source of some of this lost time. It is poor practice to allow a shortage of spare parts to cause delays.

Rigging Wire Ropes and Shore Cables.—Here is a point that is frequently overlooked in the assembling of a dredge. Where material has been transported a long distance at a high cost some erector foreman sends along a gang of workmen to rig up. They place a bundle of rope on end, and as it come off the reel a slight kink is made in each coil. This at times reduces the effective life of the rope by 50% especially in a tropical climate. An electric transmission shore cable handled the same way would be very seriously damaged and its life reduced by 50% of what it should have been. Close supervision in erection is essential for high efficiency in operating. All such material should be unwound from the reel so that it can be kept straight and free from kinks of any kind.

Opening Out from Dredge Paddocks. —Opening out should be done in a fan-shaped cut widening and deepening as the dredge advances until bedrock is reached and continues gradually widening until the maximum cut is obtained. Instances have been known where an attempt was made to reach bedrock immediately from the paddock, with the result that soon the tailing began crowding the dredge forward and room to work the dredge became so restricted that a lot of time was lost in getting out of a bad position. The best possible man should be chosen as the dredgemaster, for his skill and ability will save many delays.

Average Depths Dredged and How Determined.—Marks may be placed on the side of the digging ladder or on the ladder hoist tackle, so that the winchman may mark the average depth on each completed cut. He can amplify this by marking two or more depths on each cut across the paddock, depending on the width. From these a close average may be obtained in order to estimate the volume dug and so assist the surveyor in computing the monthly yardage treated.

Width of Cut Advisable in One Swing.—This may be determined

by practice and by no other method. The length of the hull and the ladder projection beyond the bow must be taken into consideration. It is not good practice to make the swing too far because the point between cuts becomes too sharp and this has a tendency to obstruct the step-over for beginning the adjacent cut. Experienced winchmen soon solve this problem and it need not cause any great difficulty. The better policy is not to advance too far on any forward cut before side stepping. Broad face cuts to any width are made up of a series of short arcs. Whatever is done two or three parallel cuts should not be made if such can be possibly avoided.

Gravel Pumps Off and On a Pontoon.—Under favourable conditions this method gives good results, but working costs are always higher than those for bucket-dredging under similar conditions. While the first cost of installation is less, the frequent moves, pumping out of seepage water, etc., increase operating costs. In round figures the working costs on the average of mine-run material by pumping is about five times that of bucket-dredging where conditions are suitable for each. This refers to pumps set in the paddock. On a pontoon the cost will be, perhaps, 20% less.

One of the authors finds that while theoretically the gravel pump should handle about 15% solids, that is considering the average run of material, in actual practice, allowing for power consumed and time elapsed, it will average nearer 5%. This latter figure means the average alluvial deposit, which consists of a large proportion of material of a high specific gravity. If the pumping should be from a bank 50 ft. high with 40 ft. of light wash on top, then it should handle the 15% solid.

Statements are made that, owing to more washing and cutting up of material by the impeller blades of the pump, a higher percentage extraction sounds attractive, but the economic point must be kept in mind, for a small additional saving, at a cost of more than the value of such saving, is not good mining practice. To say that the bedrock is better and more thoroughly cleaned than with the bucket-dredge is a statement open to much argument. Each method has its advantages, and every deposit is not a dredging any more than it is a gravel pumping proposition.

Reclaiming Land.—This is warranted in most workings where land has a value for farming. The designs of dredges may be various but should conform to the conditions at hand. If land has a high value then an elaborate method may be used.

Good practice is to drop the oversize from the trommel near the stern of the dredge so that it may be more easily covered by the sludge or fines. The sluices are placed at a higher elevation so that the tailing may spread more easily. In order to do this

the main gantry must be a bit higher, and the dump chute to the screen also. The hopper may have two openings, one into the screen and the other just above the top, so that a by-pass sluice may be extended from this above the screen the full length of the dredge and extend a few feet beyond all sluices. The by-pass sluice must be of a semi-circular type to give the greatest carrying capacity with a minimum of water, and liners must be provided for this sluice. When the dredge begins the cut into the top soil or overburden the lower opening to the hopper is closed and the upper opened so that this will feed the soil to the by-pass. Sufficient water must be provided to carry this material to the dump.

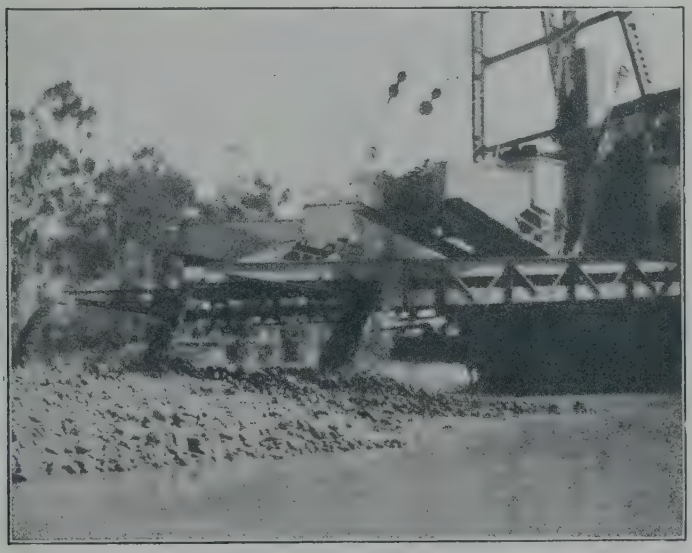

FIG. 16.—EXAMPLE OF THE PROCESS KNOWN AS "RE-SOILING".

The lower end of the sluice may be provided with a swing so that the débris may be properly distributed over the surface. This by-pass sluice can be supported from the rear gantry, extra strength being allowed for, or a belt conveyor may be used and pass the top soil dry. By use of this by-pass for soil and overburden it is possible to increase the yardage to some extent, for there is no need to wash the overburden when it carries little or no values.

Quite a bit of resoiling has been done in New Zealand and also in California and Victoria, some in California by dredgers and

others by re-levelling of the land and the removal of boulders and coarse gravel. This oversize has been crushed for concrete and railway ballast, and road making, in some instances returning a greater net profit from this source than from the gold recovery. In the early days of dredging the material was rushed through and piled in all shapes (Fig. 17), and it was also poorly washed. In the Oroville dredge areas there is some gold known to be in the débris and several schemes for its recovery and the reclaiming of this land have been under discussion. One of the best seems to be a dredge of large capacity that will drop the oversize on the bottom and place the fines on top, saving a percentage of the gold. The land has quite a good value for fruit growing. There is one instance in California where the dredged land and water rights were worth more after being worked than the company paid for it twenty years before.

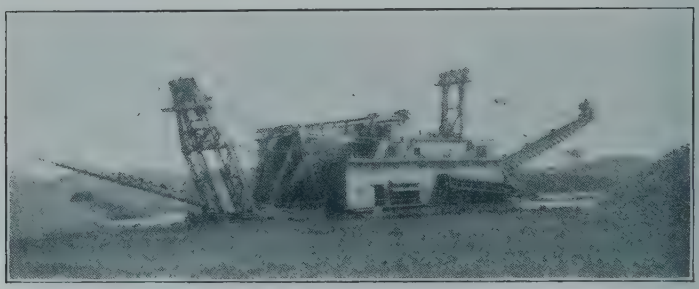

FIG. 17.—EXAMPLE OF INDISCRIMINATE HEAPING OF TAILINGS ON A DREDGE OF THE NON-RE-SOILING TYPE.

Re-working Dredged Areas.—The following are known instances that have come under observation where gravel has been re-dredged and paid more than the working cost :—One gold-bearing area was cut through that had been dredged in order to reach an unworked part. Quite a good recovery was made from this tailing, so work was continued, but as no great profit could be earned it was discontinued.

A South American platinum dredging company dug the same material three times and one of the engineers reported a profit for each operation. When asked for an explanation he said the first dredge did not reach bedrock and, the gravel being rich, it paid, so no questions were asked. The second dredge came along digging to bedrock and earned a profit. When the third was compelled to dig through this, the gravel was loose and the clay

dissolved so that the material was easily dug and more carefully washed, which also paid.

There is an instance of a tin company in Nigeria which a few years ago re-dredged some tailing and recovered more than working costs. This was to some extent due to sticky clay, and the second washing was more readily done as the operating crews had more experience.

Present practice is quite an improvement on the methods of washing the material dug as compared with the early days, but even yet perfection is not by any means attained. However, each little change for the better tends to lessen the need for re-working. Generally speaking, almost every deposit requires that the washing equipment be designed to suit the work in hand. These changes may be small and as a rule are confined to washing and sizing in the screen, sluice area, and the water supply and to the distribution of water and fines.

Dams for Stream Dredging.—At the commencement of operations in a valley with a stream flowing through it there must be some sort of temporary dam built, and this cannot be a loose earth bank because all the investment in the machinery is dependent upon this dam withstanding floods.

Experience has shewn that a dam of loose rocks thrown in and bound together by woven wire netting is very effective. If there is quite a lot of water to be held back and the boulders or rocks are in small pieces, the material may be placed on a barge and floated to the beginning point and anchored. Then bags (of a size that can be easily handled) may be made from small sheets of the wire netting doubled over, filled, sewn up with smooth annealed wire, and dumped over the side as near to the required spot as may be and until the water rises to the proper point.

If the stream has a small flow then loose rock may be thrown in until raised to the correct height and sheets of wire netting placed over the top and sewn together on the edges with wire. From these sheets wires for anchors are placed at right angles about every 30 in. apart over the whole surface, the anchor wires to be placed around the wire netting and 30 in. long. Stones are then laid on until a solid layer some 2 ft. thick has been formed, then other sheets of wire netting are laid over the stones, sewn together and tied down by means of the anchor wires, thus binding the whole into one solid sheet. The down-stream end may project several feet below lying on the surface, thus forming an apron to prevent the spill-over from undercutting the lower edge and causing a possible weakening of the structure. Grass, straw, sods, and dirt are then thrown against the upper face to make it water-tight. When this is properly done it will withstand heavy floods and provide backing for the tailing so that the dredge

may work up stream without danger from a break and leaving the hull stranded on the bottom of the paddock.

Silting of Streams from Dredging.—There is usually more or less controversy between dredge men and farmers when working in a farming country. If the facts were carefully gone into they would generally prove that the farmer is the greater offender. As an illustration, an 8 cu. ft. dredge, digging ground that averages 30 ft. deep, would turn over three acres per month or thirty-six acres per annum, less than the average farm. There is no reason to think there is more silt per acre in dredging land than in the farm. Almost every farmer cultivates more per year than the dredge turns over, and the nature of the farm lands are such that it is safe to assume that a larger amount of silt comes from the hillside farm, but the farmer always prefers to consider the mining company in some way legitimate prey : Hence the suits that are started.

Preparation of Frozen Ground.—A number of methods may be applied to the treatment of frozen ground. (*a*) The best method is to clear off the shrubbery and burn the grass and moss where possible. Other conditions being right and the valley broad and not over 30 ft. to bedrock, with little or no muck overburden, the ground will thaw with three to four years' exposure. The cost would be about ¼d. (½ cent) or less per cubic yard. (*b*) Where a plentiful water supply may be had with a good head for stripping, ground sluicing or hydraulic monitors may be used. With all conditions favourable, this has been done at a cost of ½d. (1 cent) per cubic yard, and where the average depth of bedrock is not over 40 ft. and the depth of muck averages 15 ft., it may require five years to thaw after removal of overburden. The estimates in (*a*) and (*b*) are from Klondyke practice.

The earlier method consisted of steam thawing in which long steel points were gradually driven into the ground whilst steam was forced through the pipe. The nozzle of the pipe being reinforced to withstand wear and tear and having specially designed outlets for the steam, to avoid choking with dirt whilst the driving was in progress. The process is slow and costly. The effects of steam thawing have been analysed by W. S. Weeks in U.S.A. and the results of his work show that of the total heat applied the following proportions were effective in doing useful work.

Super-heated steam 4%, saturated steam 6%, hot water 12% and cold water 57%. The temperature of the cold water was 52° F. The relative thawing efficiency in cubic yards was given as follows :—

Super-heated steam	0·69	cubic yards
Saturated steam	0·84	,, ,,
Hot water	1·20	,, ,,
Cold water	2·60	,, ,,

Cold water thawing is arranged by a layout for pipe, hose, and points in the same way as for steam, but many more will be necessary. The water requirements are a plentiful supply and a 40 ft. head at least, and it is advisable where possible to have an auxiliary high-pressure head of 100 ft. This may be obtained by a small pump for forcing the points to bedrock and then connecting the low-pressure line. The higher the temperature of the inflowing water the better. A broad ditch and low gradient is advisable. All points must be fitted with valves at the top so that the inflow may be regulated to ensure the greatest amount of heat units being extracted from the water. For instance, water may enter the points at 45° and leave the ground at 35°. The best practice

Fig. 18.—Thawing Lay-out.

is to have the water flowing up around the pipe as near to freezing point as possible. An exact temperature can only be arrived at by experience, for it is impossible for all conditions to be the same. Later practice favours cold water, i.e. about 62° F., and endeavours to circulate it at such a rate as will allow a maximum consumption of its latent heat. In other words, that the water returns as little above 32° F. as possible. The depth of deposit, temperature of the inflowing water, and other conditions must be considered to ascertain the time limits for points to remain in the ground. In average deposits the time limit is generally from 7 to 8 days. The number of points to be used must be in proportion to the volume to be thawed per day, and consideration must be given to the favourable water season. Costs

by this method will run from $1\frac{1}{2}$d. (3 cents) to 4d. (8 cents) per cubic yard and it gives about 50% efficiency.

Steam thawing is recommended for immediate results. The general layout may be judged somewhat by Fig. 18. As any two engineers will agree, few plants are the same. In a general way sufficient boiler capacity must be provided equal to the demand; also pipe, hose, and points. Larger operations are to be more carefully gone into because a tremendous amount of fuel must be

FIG. 19.—EXAMPLE OF LONG HYDRAULIC FACE IN THE KLONDYKE TO ALLOW FOR THAWING.

used. The costs per cubic yard are about 10d. (20 cents). The efficiency is about 4%. The same system may be utilized at a lower cost, approximately 6d. (12 cents) per cubic yard, with an efficiency of about 10% by using hot water instead of steam.

Several other methods have been proposed but none are to be recommended. Good general advice would be to avoid any fantastic ideas.

As regards protection against annual frost, by allowing the seasonal growth of grass and weeds to remain in the fall of the year, the early snow is easily held and it forms a protection against frost penetrating deeply. Alternatively, by an arrangement for

flooding with water when the frost begins a thin sheet of ice will soon be formed which will retain the snow, forming an insulation.

In hydraulicking, by opening out over a long face the summer heat will thaw from 10 to 24 in. per day. More equipment is required to operate by this method (Fig. 19).

Sampling Tailing.—It is recommended that the sampling of tailing should be done to a reasonable extent to check losses and to determine if the dredge crews have been digging to bedrock. Some years ago, in one California district, large yardages were demanded, and quite a rivalry developed between a number of dredge crews. The depth to bedrock being known, some crews reported bedrock and depths. Later some re-boring of tailing was done and it transpired that a few feet of the bottom gravel and bedrock had not been worked, thus leaving behind the best values. After careful re-boring and estimating it was found that it would not be advisable to re-dredge these areas. This is one detail that requires close attention, for winchmen are only human beings.

Mine Timbers and Buried Trees.—Old mine timbers, where extensive work has been done, are a source of trouble and must be reckoned with in any district. While tests were being made in the Cariboo country, British Columbia, during 1923 and 1924, it was found that some sawn timbers, water-logged for thirty years, would sink on being freed from the ground, but that round timbers would float with a small portion showing. When the sawn timbers were broken, however, they would float in the same way as the round timbers, with the exception that the broken end would show a little above the water. The conclusion drawn was that these timbers on being cut or torn by a powerful dredge would more or less float, and could then be removed from the paddock if they should become an obstruction to the operations. About $\frac{1}{4}$d. ($\frac{1}{2}$ cent.) per cubic yard was added to the estimated cost for this. There should be a small credit for fuel value from these timbers after they become seasoned. The amount will be much greater in the north, especially in places like the Lena Mines in Siberia, where the fuel supply is quite an important item. Buried trees have been found to be quite a handicap in the hard-wood countries, especially in the tropics, and a great deal of trouble was caused on the Ropp Mine in Nigeria from this cause. About the best way to handle these is to expose as much of the log as may be convenient, place a good charge of dynamite against it (under water is preferable) swing the dredge some distance away as a precaution against damage, and then fire the shot. One such will generally break the log so that it may be torn out and removed by the light winch.

Dredging Stream Channels Only.—It is practical and profitable

to dredge stream channels only when deposits are suitably situated and where the depth of water and gravel is sufficient to give flotation for working the dredge, and there are found concentrations that are workable. These are generally due to the streams having cut away several feet of the overburden and the metals have been concentrated in the few feet of material lying upon bedrock, which enrichment in many places is being increased during every flood period. Such deposits will not all carry payable values, but the bars on the short side of the bends must prove the best.

About 1911 on the Ancobra River, West Africa, the practice of working only these bars gave fair returns for a time, but were handicapped owing to lack of water at other times. Generally this would not be possible but only on streams that have a great many tributaries bringing down values to afford sufficient enrichment.

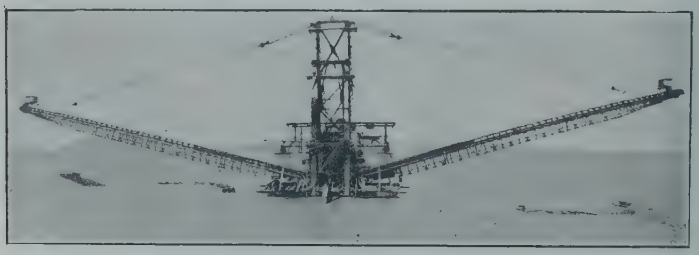

FIG. 20.—WIDE DOUBLE STACKER FOR LEAVING OPEN CHANNEL WHEN DREDGING STREAMS.

Glacial Deposits for Dredging.—From the experience of one of the authors extending over several years, it is concluded that it is almost impossible to find a profitable proposition in glacial deposits. Several have been tried in Canada, the United States and Alaska, and none has paid real profits. Such deposits do not in their manner of forming ever seem favourable, owing to the terminal moraine masses of boulders, but below such deposits—where large streams have rearranged and concentrated the material—it is a possible place to prospect. The glacier is often a useful crushing plant to prepare the material and free the metals.

Dredging Material Containing Boulders.—Little preparation is required for this kind of deposit. If any boulders are on top or partly exposed it is best to break them up with explosives at once. Open-connected bucket lines of large size seem the logical thing, but it has been found good practice to change the design

of the bucket itself and use the closed line. This design is indicated in Fig. 10, page 123. Being cut well back on the digging side, this allows more room between the lip of one bucket and the hood of the next, thus preventing breakages.

Value of Tests and Safety Factor.—The value of tests is as a rule equal to the experience that the engineer has had. If he should ignore all schemes and methods that have been worked out or used before, then his results may be taken with a certain reservation, because there is some good in every scheme and a combination of the good points is recommended. When tests are carefully made, and by the approved practice, they may be taken as proved facts. The safety factor that is here advised has been proved correct, not theoretically or by checking a few boreholes by shafts, but by the dredging of some 500,000,000 cubic yards of ground after boring.

At one time 100 linear feet of 6 inch pipe was taken to equal one cubic yard. The same sized pipe with a 7·5 inch shoe (outside diameter) works out at 88 linear feet, but in practice it has been found that 81 linear feet is more nearly the correct length with that sized shoe.

Formula.

If C = Money value.
D = Depth in ft. or m.
L = Length of cut or cylinder to 1 cu. yd. or cu. m.
V = Value per cu. yd. or cu. m.

$$\text{Then } V = \frac{C + L}{D}$$

Recovery of Broken Bucket Line.—This is one of the most difficult repair jobs on a dredge and a cause of lost time. Nothing more serious can happen, except the sinking of the pontoon. Once the line is submerged it is more or less guesswork to try and make fast to the proper part of the broken line without the use of a diver. A diver should always be sent down, with or without a diving suit. It is almost impossible to recover a line from one of the larger sized dredges with the ordinary hook, as a catch of the lower part of the line might involve the raising of 300 tons. This is more easily understood when it is remembered that all the work is done in muddy water.

Raising a Sunken Dredge.—Fig. 21 shows one of the 17 cu. ft. dredges of the Canadian Klondyke after she sank in the channel of the Klondyke River. To raise, repair and float this cost over £10,000 ($50,000) and took several months' time. First it was necessary to divert the stream and then pump out the paddock. As there was quite an area of worked ground behind the dredge, and as this was somewhat porous, it required a lot of pumping

to clear out the accumulated water. Once the water was down the work was not so great. In addition to the above cost there was the loss of one productive season; this in round figures indicated a net loss of profits of £40,000 ($200,000). The foregoing is only one of many, and it was the most expensive that is on record. It is the best practice where possible to call in professional salvage crews to refloat a sunken dredge.

Rebuilt Dredges.—Rebuilding a dredge is advisable where all conditions are favourable. The purchase of good second-hand machinery near some line of transport is recommended as a rule, the freight per ton on this class of material being the same as that on new machinery. Seldom is the steel or wood of an old hull worth moving, other than the frame work. Such rebuilt dredges have been successful in Nigeria, Klondyke, Alaska, the United

Fig. 21.—Klondyke No. 2 Wrecked.

States and Siberia. In the purchase of such machinery it is best to get someone well posted in that class of material to make an examination and appraise it before the purchase is completed. Several times these deals have been carried out without this advice and the rebuilt dredge cost more than a new one would have.

Winter Quarters for the Dredge in Cold Countries.—This is a point that wants every consideration, for if the work is carelessly done it may mean a considerable delay in the spring start. If the hull be of wood then all caulked seams along the sides of the hull must be covered with good battens well fastened on. In the beginning of dredging in the Klondyke it was the practice to lower the water in the paddock excavate a shallow connected paddock with a level bottom, then raise the water and float the dredge into this, afterwards opening a drain and lowering the water, leaving the hull on the bottom. Small wood hulls were

in use. Subsequent practice has been to have a good depth of water under the hulls and leave them in the paddock. This method has proved satisfactory and is now generally in use.

ADMINISTRATION.

Management and Staff.—The manager and his assistant should be men with experience in this class of mining. Having a man of this kind, it is best to place the selection of the staff entirely in his hands, for if the directors of the company make such selection, it sometimes leads to friction, because men so selected get the idea that they can go over the manager's head direct to the board. Care must also be used to get efficient, reliable men, even at a small advance in salary. It will prove a splendid investment in the end. It is well to work out a reasonable advance to be allowed in annual payments. It is always best to retain the same staff whenever possible. The experienced manager will not load up the business with superfluous staff.

The general operation should be under the direction of the consulting or superintending engineer with the board's approval over certain parts.

No attempt should be made to allow any crew to work on a contract per cubic yard, or to pay a bonus for excess yardage. This will lead to slighting the work, poor recovery, etc. The man who proposes to increase the output if his pay is raised knows he is not doing his duty but hopes the manager does not, and he should be sacked at once.

Safety Rules.—These should be conspicuously posted at several points on the dredge, and they should cover every possible danger. To assist in the observation of these, all possible safety devices must be used, and explicit orders given that these are to be kept in place and in good working condition. Slackness on this point is likely to cause carelessness on the part of the crew.

Camps and Quarters.—These are sadly neglected in many mining operations. In the exploratory stages, almost any camp will do, but when it is known that work is to continue, comfortable camps and living quarters should be provided. Many companies operating in out-of-the-way places get a bad name because of their camps, and this makes it very difficult to get the best men to accept positions. Tropical zones require more care in the selection of camp sites and the erection of quarters. The top of a hill or ridge above the ordinary level or floor of the valley should be chosen, and all grass and brush cut away, including the lower branches of shade trees. It is best to have only a few shade trees. This clearing should be about 200 yards square. Too much shade will harbour mosquitoes, which insects must be eliminated if

possible, and against which all quarters should be well screened. Comfortable quarters mean a contented staff and this begets high efficiency.

Report Sheets.—It is well known that some operations in this branch of the business are overdone. There should be sufficient forms so that each cost item may be allocated, and an analysis made in order to check expenditure and no more. The following will be found ample, with slight modifications, to cover different conditions :—

ACCOUNT NUMBERS.

Running Expenses.	*Supplies Used.*
1. Material.	*a.*
2. Power.	*b.*
3. Labour.	*c.*
4. Water.	*d.*
Repairs.	*e.*
5. Material.	*f.*
6. Labour.	*g.*
General Expense.	*h.*
7. Fire protection.	*i.*
8. Accident Insurance.	*j.*
9. Levee.	*Machine Shop.*
10. Restraining débris.	*a.*
General Items.	*b.*
11. Insurance.	*c.*
12. Taxes.	*d.*
13. Depreciation.	*e.*
14.	*Shift Sheets.*
15.	(These should contain lines enough for a clear statement to be made.)

In general, sheets for each class of dredging should differ; one for gold operations is not quite suitable for tin. Various coloured papers are advisable as an aid to filing.

Monthly Volume Plans.—The original monthly volume plans (Fig. 22) should always be kept at the field office and a copy at the home office. Then, as weekly or monthly returns are entered on the original, small tracings may be forwarded to be entered on the home office plan, thus keeping it closely up to date. This is of particular value when the head office is distantly removed from the scene of operations.

COSTS.

Cost Details and Graphs.—Detailed costs are a most important item and should be carefully worked out from the shift, daily and monthly report forms. Records in some form should always be kept, and though small scale operations may not justify an elaborate system of "costing", a close watch on these details will often effect considerable economies. The main object is to accomplish a high recovery and the next step is to reduce costs as low as possible.

Bucket Dredging Costs.—Operating or field costs per cubic yard for gold and platinum vary from 6d. to 10d. on a dredge with 3 cu. ft. buckets, and from $1\frac{1}{2}$d. to 7d. on a dredge of 17 cu. ft. capacity. Intermediate figures may be approximately arrived at from these rates. They will vary from month to month on the same area ; for tin the field costs on a 3 cu. ft. dredge range from 7d. to 1s. per cubic yard and on a dredge of 10 cu. ft. capacity from 3d. to 7d. per cubic yard. All costs vary in the same field, while quite a difference will be found in the comparative costs on various other fields.

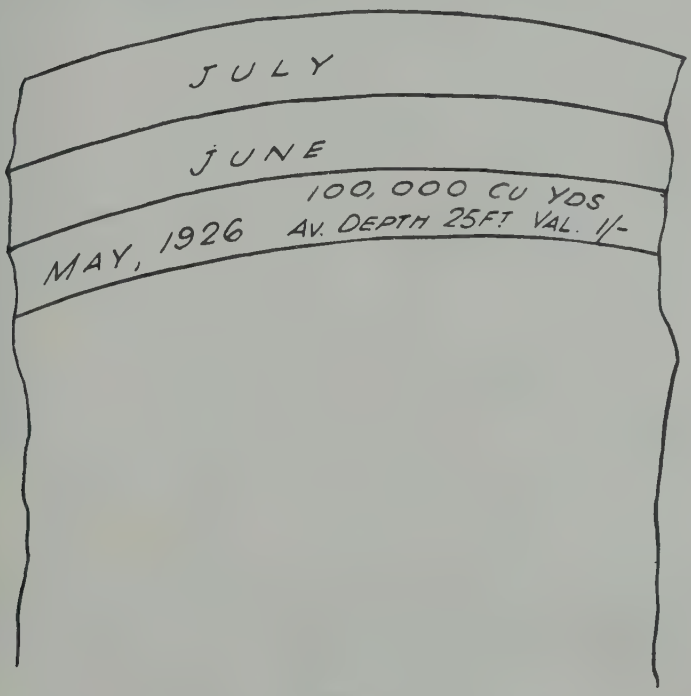

FIG. 22.—MONTHLY VOLUME PLAN.

There are no accurate costs to be obtained at the present time for work on diamonds, but from experience 8s. per cubic yard may be assumed as a safe figure for estimates. Large sized dredges are not practical for working diamond-bearing ground.

Operating Costs.—These vary in almost every country and at times in the same deposit on the same dredge, due to many conditions. A few are mentioned that are worth considering, with the object of avoiding or overcoming them.

Material.—This should always be the best. Though first cost may be a little more, the cost of transport is the same ; and there is a greater saving in repairs and renewals, thus keeping the lost time chargeable at a minimum. A plentiful supply of spares must always be at hand, not in excess, but enough. In isolated districts larger stocks are essential. Under rare circumstances is a poor quality warranted. Lighter material under certain conditions may be stocked, such as lips, bushings, etc. This was

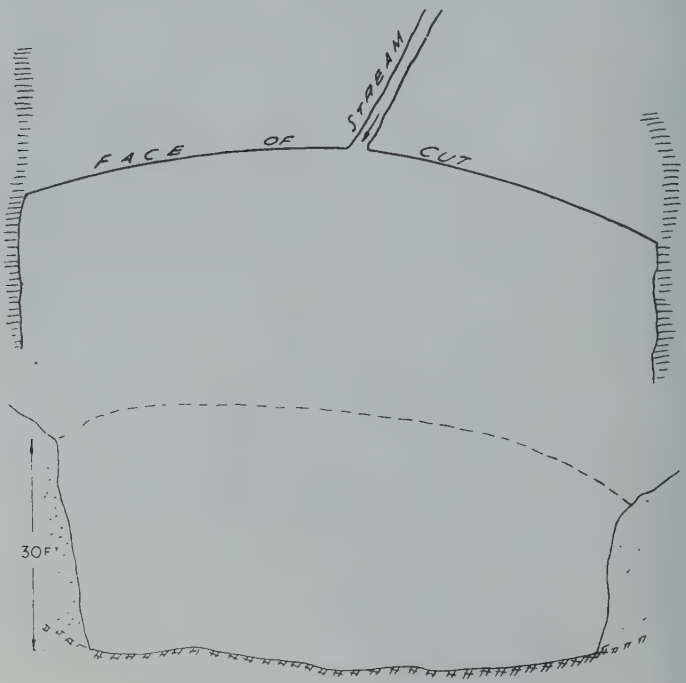

Fig. 23.—Plan and Cross Section of Single Cut.

found good practice on the Canadian Klondyke a few years ago, each productive working season being about 7 to 8 months, so that weight was reduced in such wearing parts that were renewed at the close of each season. This gave a better running time, for otherwise they were obliged to scrap parts that were perhaps only from 75 to 80% worn.

Width of Cut.—By keeping this at a maximum costs will be less (see Fig. 23).

Setting "Dead Men" or Anchors.—These require a set of each for every cut, as shown in Fig. 24. There is not only the possibility of bedrock not being well cleaned but it is necessary to re-handle some of the tailings for the angle is less in loose material. Assuming ground to be 30 ft. deep as shown in Fig. 24, then for each return cut and each foot of advance 33 cubic yards are re-dredged or 66 cubic yards for a three-cut width, besides losses in the wedged shaped sections on the bedrock that are left behind. Work comprised in setting dead men is not included.

Percentage Costs.—The following table gives about the average of operating costs under favourable conditions, for a 7 or 8 cu. ft. dredge in the temperate zone on gold or platinum.

	Pence per cubic yard.
Labour	0·6
Power	0·4
Repairs and supplies	0·8
Taxes and insurance	0·2
Miscellaneous	0·2
Amortization	0·4
Total	2·6

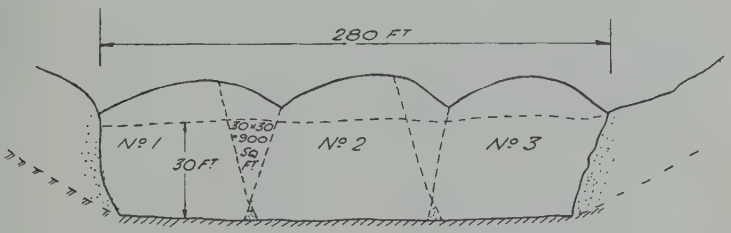

FIG. 24.—CROSS SECTION OF THREE CUTS.

The percentages are about as follows: Material and repairs about 30%, labour, power, etc., about 70%. In the tropics the percentages will increase on an average about 30% and under Arctic conditions from 50% to 200%. The condition of the deposits govern the latter, these being frozen and in more inaccessible districts.

The following tabulation is for a 17 cu. ft. dredge operating in the Klondyke:—

	Pence per cubic yard.
Labour	0·16
Power	0·75
Amortization	1·49
Repairs	0·65
Management	1·25
Total	4·30

The above figures are given for the purpose of assisting in estimates and may be taken as accurate at the time and place from which they are reported.

In 1917 H. D. Griffiths wrote in *The Mining Magazine* of tin dredging costs in the Malay States as follows :—

Dredging Costs :—	Pence per cubic yard.
Labour	0·72
Fuel	0·25
Maintenance	0·12
Stores	0·21
Renewals	0·16
Total	1·46

Dressing Costs :—	Pence per cubic yard.
Labour	0·16
Stores	0·02
Sundries	0·03
Total	0·21

From these figures it would appear that the work was carried out under ideal conditions and includes only actual field costs, no allowance having been made for depreciation.

Bucket Dredge Construction Costs.—In making an estimate of these costs a safe figure as an average may be taken at £60 ($300) per ton for manufacture and erection plus freight.

Three sizes are given in the following table :—

Capacity. Cu. ft.	Wt. of Machinery for 30 ft. depth. Tons.	Wt. Wood Hull. Tons.	Wt. Steel Hull. Tons.
4	180	275	155
8	360	530	340
18	1,000	1,000	900

These weights will not be the same for all designs as they vary slightly. For an open line they will be less. Intermediate sizes may be worked out from the figures given. The size of the dredge required may be arrived at by the following figures :—

Capacity. Cu. ft.	Theoretical cu. yds. 24 hours.	Actual Cubic Yards.
4	5,200	3,000
8	9,500	5,000
18	20,000	13,400

The life of a property is based on the time during which it is desired to operate, generally from 10 to 12 years. The above costs are for the best type of design and high grade material. With poor material, etc., the costs may be less by 20%.

APPENDIX.

Table for Estimating Approximately the Quality of Tin Ore by Sp. Gravity.

This table is compiled only on the assumption that the impurity present is iron and is of no use when sand is present in any considerable quantity. It is fairly accurate up to 1·5% silica.

To use the Table :—Take 126 grammes or 1944·4 grains of the sample. Fill a burette with a known quantity of water, add the sample and note the displacement. Then read the corresponding value in the table, which is given in the percentage of black tin, and metallic tin.

Example :—Displacement 23 cc's, % Tin Oxide 50%, $50 \times 0{\cdot}787 = 39{\cdot}35\%$ Metallic Tin.

Table.

Displacement in cc.'s.	Black Tin.	Metallic Tin.	Displacement in cc.'s.	Black Tin.	Metallic Tin.	Displacement in cc.'s.	Black Tin.	Metallic Tin.
18·0	100	78·7	19·0	90	70·8	20·0	80	62·9
18·1	99	77·9	19·1	89	70·04	20·1	79	62·17
18·2	98	77·1	19·2	88	69·27	20·2	78	61·38
18·3	97	76·35	19·3	87	68·47	20·3	77	60·6
18·4	96	75·5	19·4	86	67·69	20·4	76	59·8
18·5	95	74·7	19·5	85	66·9	20·5	75	59·02
18·6	94	73·9	19·6	84	66·1	20·6	74	58·2
18·7	93	73·1	19·7	83	65·3	20·7	73	57·4
18·8	92	72·4	19·8	82	64·5	20·8	72	56·6
18·9	91	71·6	19·9	81	63·7	20·9	71	55·8
21·0	70	55·09	23·3	47	36·99	25·6	24	18·89
21·1	69	54·3	23·4	46	36·2	25·7	23	18·1
21·2	68	53·5	23·5	45	35·4	25·8	22	17·3
21·3	67	52·7	23·6	44	34·6	25·9	21	16·5
21·4	66	51·9	23·7	43	33·8	26·0	20	15·7
21·5	65	51·1	23·8	42	33·0	26·1	19	14·95
21·6	64	50·3	23·9	41	32·26	26·2	18	14·1
21·7	63	49·5	24·0	40	31·4	26·3	17	13·38
21·8	62	48·7	24·1	39	30·69	26·4	16	12·59
21·9	61	48·0	24·2	38	29·9	26·5	15	11·8
22·0	60	47·2	24·3	37	29·1	26·6	14	11·01
22·1	59	46·4	24·4	36	28·3	26·7	13	10·2
22·2	58	45·6	24·5	35	27·5	26·8	12	9·4
22·3	57	44·8	24·6	34	26·7	26·9	11	8·6
22·4	56	44·0	24·7	33	25·9	27·0	10	7·87
22·5	55	43·2	24·8	32	25·1	27·1	9	7·07
22·6	54	42·40	24·9	31	24·4	27·2	8	6·8
22·7	53	41·7	25·0	30	23·6	27·3	7	5·5
22·8	52	40·9	25·1	29	22·8	27·4	6	4·7
22·9	51	40·1	25·2	28	22·0	27·5	5	3·9
23·0	50	39·35	25·3	27	21·25	27·6	4	3·14
23·1	49	38·5	25·4	26	20·4	27·7	3	2·36
23·2	48	37·7	25·5	25	19·68	27·8	2	1·57
			25·6	24	18·89	27·9	1	0·78

To facilitate transport by native carriers it has been the practice in Nigeria to make up all material in loads approximating 70 lb. Tin ore is done up in double canvas sacks containing 70 lb. nett or 32 bags to the ton. The following table has been prepared to assist operators in calculating results to the nearest bag and expressing same in tons and decimals of a ton as required by the Nigerian Mines Department in the official returns which every operator must make monthly:

Bags of Tin.	Pounds weight.	Dec. of long ton.	
1	70	·03125	32 bags = 1 ton.
2	140	·0625	64 ,, = 2 ,,
3	210	·09375	96 ,, = 3 ,,
4	280	·125	128 ,, = 4 ,,
5	350	·15625	160 ,, = 5 ,,
6	420	·1875	192 ,, = 6 ,,
7	490	·21875	224 ,, = 7 ,,
8	560	·250	256 ,, = 8 ,,
9	630	·28125	288 ,, = 9 ,,
10	700	·3125	320 ,, = 10 ,,
11	770	·34375	480 ,, = 15 ,,
12	840	·375	640 ,, = 20 ,,
13	910	·40625	800 ,, = 25 ,,
14	980	·4375	960 ,, = 30 ,,
15	1050	·46875	1120 ,, = 35 ,,
16	1120	·500	1280 ,, = 40 ,,
17	1190	·53125	1440 ,, = 45 ,,
18	1260	·5625	1600 ,, = 50 ,,
19	1330	·59375	1760 ,, = 55 ,,
20	1400	·625	1920 ,, = 60 ,,
21	1470	·65625	2080 ,, = 65 ,,
22	1540	·6875	2240 ,, = 70 ,,
23	1610	·71875	2400 ,, = 75 ,,
24	1680	·750	2560 ,, = 80 ,,
25	1750	·78125	2720 ,, = 85 ,,
26	1820	·8125	2880 ,, = 90 ,,
27	1890	·8435	3040 ,, = 95 ,,
28	1960	·875	3200 ,, = 100 ,,
29	2030	·90625	
30	2100	·9375	
31	2170	·96875	
32	2240	1·000	

TABLE SHOWING TONS AND FRACTIONS OF A TON EXPRESSED IN LB. AVOIRDUPOIS.

Tons.	Ton in lb.	¼ ton.	½ ton.	¾ ton.	Tons.	Ton in lb.	¼ ton.	½ ton.	¾ ton.
1	2240	2800	3360	3920	14	31360	31920	32480	33040
2	4480	5040	5600	6160	15	33600	34160	34720	35280
3	6720	7280	7840	8400	16	35840	36400	36960	37520
4	8960	9520	10080	10640	17	38080	38640	39200	39760
5	11200	11760	12320	12880	18	40320	40880	41440	42000
6	13440	14000	14560	15120	19	42560	43120	43680	44240
7	15680	16240	16800	17360	20	44800	45360	45920	46480
8	17920	18480	19040	19600	25	56000	55000	123200	
9	20160	20720	21280	21840	30	67200	60000	134400	
10	22400	22960	23520	24080	35	78400	70000	156800	
11	24640	25200	25760	26320	40	89600	80000	179200	
12	26880	27440	28000	28560	45	100800	90000	201600	
13	29120	29680	30240	30800	50	112000	100000	224000	

Comparisons and Equivalents.

Troy Weight.

For weighing precious metals, such as gold, platinum, silver, etc., Troy Weights are used exclusively.

The U.S. Standard Troy pound was copied in 1827 from the Imperial Troy pound of England for the use of the United States Mint, and there deposited. It is standard in air at 62° Fahr., the barometer at 30 inches.

24 grains = 1 dwt.
480 ,, = 20 ,, = 1 oz.
5,760 ,, = 240 ,, = 12 ,, = 1 lb. = 22·816 cu. in. of distilled water at 62° Fahr.

Avoirdupois Weight.

For weighing base metals such as lead, antimony, tin, etc., and the weight in general use in trade and commerce.

1 drachm = 27·34375 grains Troy.
16 ,, = 1 oz. = 437·5 ,, ,,
256 ,, = 16 ,, = 1 lb. = 1·2153 lb. Troy.
6,400 ,, = 400 ,, = 25 ,, = 1 quarter.
25,600 ,, = 1,600 ,, = 100 ,, = 4 ,, = 1 cwt. } shrt.
512,000 ,, = 32,000 ,, = 2,000 ,, = 80 ,, = 20 ,, = 1 ton } ton.

Apothecaries' Weight.

20 grains = 1 scruple.
60 ,, = 3 ,, = 1 drachm.
480 ,, = 24 ,, = 8 ,, = 1 oz.
5,760 ,, = 288 ,, = 96 ,, = 12 ,, = 1 lb.

Metric or French Weights.

	Grammes.	grs. Troy.	Troy. oz.	Troy. lb.	Avoir. oz.	Avoir. lb.
1 Milligramme =	0·001 =	·01543				
1 Centigramme =	0·01 =	·15432				
1 Decigramme =	0·1 =	1·5432				
1 Gramme . =	1 =	15·432 =	0·032 =	0·00267 =	0·03528 =	0·0022047
1 Decagramme =	10 =	— =	0·321 =	0·02679 =	0·3528 =	0·022046
1 Hectogramme =	100 =	— =	3·215 =	0·26792 =	3·52758 =	0·22046
1 Kilogramme =	1,000 =	— =	32·150 =	2·6792 =	35·2758 =	2·2046
1 Myriagramme =	10,000 =	— =	— =	26·792 =	— =	22·046
1 Quintal . =	100,000 =	— =	— =	267·92 =	— =	220·46
1 Tonneau . =	1,000,000 =	— =	— =	2679·2 =	— =	2204·6

The unit of the metric system is the gramme = 15·438395 Troy grains, or the weight of 1 c.c. of distilled water at 4° C.

Assay Ton Weights.

The Assay Ton Weight is a system made up from a comparison of the Avoirdupois, Troy and Gramme Weights, and will be found extremely simple and useful, saving a vast amount of calculation and labour.

The Unit of the system is the Assay Ton (A.T.) = 29·166 grammes. Its derivation will be seen at a glance : (*a*) Short ton, (*b*) the Long ton.

(*a*) 1 lb. Avoirdupois = 7,000 Troy grains.
2,000 lb. = 1 ton.
2,000 × 7,000 = 14,000,000 Troy grains, in one ton Avoirdupois.
480 Troy grains = 1 oz. Troy.
14,000,000 ÷ 480 = 29,166 Troy ozs. in 2,000 lb. Avoirdupois.
There are 29,166 milligrammes in one assay ton (A.T.); hence 2,000 lb. is to 1 A.T. as 1 oz. Troy is to 1 milligramme.
Therefore if 1 A.T. of ore assays 1 milligramme of gold or silver, the ton contains 1 ounce Troy.

(*b*) For the long ton 15,680,000 Troy grains in 1 ton Avoirdupois.
15,680,000 ÷ 480 = 32·666 Troy ozs. in 2,240 lb. Avoirdupois.
There are 32·666 milligrammes in one assay ton; hence 2,240 lb. is to one A.T. as 1 oz. Troy is to 1 milligramme.
Therefore if 1 A.T. of ore assays 1 m.g. the ton contains 1 oz. Troy.

Table to convert Metric Weights into Avoirdupois and Troy Weights.

As one gramme is equal to 15·432 × grains, or 0·03527 Avoirdupois ounce, or 0·03215 Troy ounce to convert :—

Grammes .	into grains	multiply by	15·432
Centigrammes	,, ,,	,,	,, 0·15432
Milligrammes	,, ,,	,,	,, 0·01543
Kilogrammes	,, Avoirdupois ozs.	,,	,, 35·2739
Grammes .	,, ,, ,,	,,	,, ·03527
Kilogrammes	,, ,, pounds	,,	,, 2·2046
Kilogrammes	,, Troy ounces	,,	,, 32·1507
Grammes .	,, ,, ,,	,,	,, ·03215

Comparison of Avoirdupois, Metric, Troy and Assay Ton Weights.

	Avoir. Ounce.	Avoir. Pound.	Milli-grammes.	Centi-grammes.	Deci-grammes.	Grammes.	Deca-grammes.	Hecta-grammes.	Kilo-grammes.	Troy Grain.
1 Avoir. Oz.	1	·06250	28,349·5403	2,834·95403	283·495403	28·3495403	2·83495403	0·283495403	0·0283495403	437·5
1 ,, lb.	16	1	453,592·6449	45,359·26449	4,535·926449	453·5926449	45·3592449	4·535926449	0·4585926449	7,000
1 Milligramme	0·00003527394	0·00000220462	1	0·1	0·01	0·001	0·0001	0·00001	0·000001	0·015432349
1 Centi	0·0003527394	0·0000220462	10	1	0.1	0·01	0·001	0·0001	0·00001	0·15432349
1 Deci	0·003527394	0·000220462	100	10	1	0·1	0·01	0·001	0·0001	1·5432349
1 Gramme	0·03527394	0·00220462	1,000	100	10	1	0·1	0·01	0·001	15·432349
1 Decagramme	0·3527394	0·0220462	10,000	1,000	100	10	1	0·1	0·01	154·32349
1 Hecta	3·3527394	0·220462	100,000	10,000	1000	100	10	1	0·1	1,543·2349
1 Kilo	35·27394	2·20462	1,000,000	100,000	10000	1,000	100	10	1	15,432·349
1 Troy Grain	0·00228571	0·000142857	64·79897	6·479897	0·6479897	0·06479897	0·006479897	0·0006479897	0·00006479897	1
1 ,, Dwt.	0·0548571	0·0034285	1,555·1754	155·51754	15·551754	1·5551754	0.15551754	0·015551754	0·0015551754	24
1 ,, Ounce	1·0971428	0·0685714	31,103·495	3,110·3495	311·03495	31·103495	3·1103495	0·31103495	0·031103495	480
1 ,, Pound	13·165714	0·822857	373,241·9478	37,324·19478	3732·419478	373·2419478	37·32419478	3·732419478	0·3732419478	5,760
1 Assay Ton	1·0288232	——	——	——	——	29·1666666	——	——	——	——

Table to convert Avoirdupois and Troy Weights into Metric Weights.

As one grain is equal to 0·0648 and one Avoirdupois ounce is equal to 28·3495 grammes, and one Troy ounce is equal to 31·1035 grammes; to convert:—

Grains	into grammes	multiply by	0·0648
Grains	,, centigrammes	,,	,, 6·4799
Grains	,, milligrammes	,,	,, 64·799
Avoirdupois ozs.	,, kilogrammes	,,	,, 0·02835
Avoirdupois ozs.	,, grammes	,,	,, 28·3495
Avoirdupois pounds	,, kilogrammes	,,	,, 0·4536
Troy ounces	,, kilogrammes	,,	,, 0·0311
Troy ounces	,, grammes	,,	,, 31·1035

Linear Measure.

Inches.	Foot.	Yard.	Fathom.	Perch.	Furlong.	Mile.	
12 =	1						
36 =	3 =	1					
72 =	6 =	2 =	1				
198 =	16·5 =	5·5 =	2·75 =	1			
7,920 =	660 =	220 =	110 =	40 =	1		
63,360 =	5,280 =	1,760 =	880 =	320 =	8	= 1	League.
190,080 =	15,840 =	5,280 =	2,640 =	960 =	24	= 3	= 1

Linear Measure, Metric.

	Metre.	Ins.	Feet.	Yards.	Miles.
1 Millimetre =	0·001 =	0·03937 =	0·00328		
1 Centimetre =	0·01 =	0·3937 =	0·0328		
1 Decimetre =	0·1 =	3·937 =	0·32808 =	0·10936	
1 Metre =	1 =	39·3704 =	3·2808 =	1·0936	
1 Decametre =	10 =	393·704 =	32·808 =	10·936	
1 Hectometre =	100 =	— =	328·08 =	109·36	= 0·0621375
1 Kilometre =	1,000 =	— =	3,280·8 =	1,093·6	= 0·621375
1 Myriametre =	10,000 =	— =	32,808 =	10,936	= 6·21375

Table to convert Linear Measure into Metric Linear Measure.

As one inch is equal to 0·0254 metres; to convert:—

Inches	into metres	multiply by	0·0254
,,	into centimetres	,,	,, 2·5399
,,	into millimetres	,,	,, 25·3997

Table to convert Metric Linear Measure into Imperial Linear Measure.

As one metre is equal to 39·370 inches; to convert:—

Metres	into inches	multiply by	39·370
Centimetres	,, ,,	,,	,, 0·3937
Millimetres	,, ,,	,,	,, 0·03937

Cubic Measure.

1,728 cubic inches = 1 cubic foot.
46,656 cubic inches = 27 cubic feet = 1 cubic yard.
A cubic foot of water weighs 62½ pounds, and contains 1,728 cubic inches, or 6¼ gallons.

Cubic Measure, Metric.

	Cu. Metres.	Cu. Ins.	Cu. Ft.	Cu. Yds.
1 Cubic Centimetre =	0·000001 =	0·061025		
1 Cubic Decimetre =	0·001 =	61·025		
1 Centistere =	0·01 =	610·25 =	0·353156	
1 Decistere =	0·1 =	6102·5 =	3·53156 =	0·13080
1 Stere =	1 =	— =	35·3156 =	1·3080
1 Decastere =	10 =	— =	353·156 =	13·080
1 Hectostere =	100 =	— =	3531·56 =	130·80

Square Measure.

Inches.	Foot.	Yard.	Perch.	
144 =	1			
1,296 =	9 =	1		
39,204 =	272·25 =	30·25 =	1	
1,568,160 =	10,890 =	1,210 =	40	Acre.
6,272,640 =	43,560 =	4,840 =	160 =	1

Square Measure, Metric.

	Sq. Metres.	Imp. Sq. In.	Sq. Ft.	Sq. Yards.	
1 Sq. Centimetre =	0·0001 =	0·155 =	0·10764 =	0·01196	Acres.
1 Sq. Decimetre =	0·01 =	15·5 =	0·10764 =	0·01196	Acres.
1 Centiare =	1 =	1,550·03 =	10·764 =	1·196	= 0·00024
1 Are =	100 =	155,003 =	1,076·4 =	119·6	= 0·0247
1 Hectare =	10,000 =	— =	107,641 =	11,960	= 2·47

Table for reducing Square Feet to Acres.

Square Feet.	Acres.	Square Feet.	Acres.	Square Feet.	Acres.	Square Feet.	Acres.
100,000,000	2,295·684						
90,000,000	2,066·116	900,000	20·661	9,000	·207	90	·0021
80,000,000	1,836·547	800,000	18·365	8,000	·184	80	·0018
70,000,000	1,606·979	700,000	16·070	7,000	·161	70	·0016
60,000,000	1,377·410	600,000	13·774	6,000	·138	60	·0014
50,000,000	1,147·842	500,000	11·478	5,000	·115	50	·0011
40,000,000	918·274	400,000	9·183	4,000	·092	40	·0009
30,000,000	688·705	300,000	6·887	3,000	·069	30	·0007
20,000,000	459·137	200,000	4·591	2,000	·046	20	·0005
10,000,000	229·568	100,000	2·296	1,000	·023	10	·0002
9,000,000	206·612	90,000	2·066	900	·021	9	·00021
8,000,000	183·655	80,000	1·836	800	·018	8	·00018
7,000,000	160·698	70,000	1·607	700	·016	7	·00016
6,000,000	137·741	60,000	1·377	600	·014	6	·00014
5,000,000	114·784	50,000	1·148	500	·011	5	·00011
4,000,000	91·827	40,000	·918	400	·009	4	·00009
3,000,000	68·870	30,000	·689	300	·007	3	·00007
2,000,000	45·914	20,000	·459	200	·005	2	·00005
1,000,000	22·957	10,000	·230	100	·0023	1	·00002

TABLE FOR ESTIMATING THE VALUE OF SILVER PER TROY OUNCE at Different Degrees of Fineness, based on $0·50 per Ounce 1,000 Fine.

To find the present market value of silver at any given time, add 2% for every 1 c. above 50 c. Example: To estimate the value of 1 ounce of silver 750 fine, presuming the market value of silver is 63 c. per ounce for silver 1,000 fine, which is 13 cents, or 26% above the computed value in table below, thus: One ounce of silver 750 fine is worth $37\frac{1}{2} \times 26\% = 47\frac{1}{4}$ c.

Fine.	Cents.	Fine.	Cents.	Fine.	Cents.	Fine.	Cents.
10	00·50	260	13	510	25·50	760	38
20	01	270	13·50	520	26	770	38·50
30	01·50	280	14	530	26·50	780	39
40	02	290	14·50	540	27	790	39·50
50	02·50	300	15	550	27·50	800	40
60	03	310	15·50	560	28	810	40·50
70	03·50	320	16	570	28·50	820	41
80	04	330	16·50	580	29	830	41·50
90	04·50	340	17	590	29·50	840	42
100	05	350	17·50	600	30	850	42·50
110	05·50	360	18	610	30·50	860	43
120	06	370	18·50	620	31	870	43·50
130	06·50	380	19	630	31·50	880	44
140	07	390	19·50	640	32	890	44·50
150	07·50	400	20	650	32·50	900	45
160	08	410	20·50	660	33	910	45·50
170	08·50	420	21	670	33·50	920	46
180	09	430	21·50	680	34	930	46·50
190	09·50	440	22	690	34·50	940	47
200	10	450	22·50	700	35	950	47·50
210	10·50	460	23	710	35·50	960	48
220	11	470	23·50	720	36	970	48·50
230	11·50	480	24	730	36·50	980	49
240	12	490	24·50	740	37	990	49·50
250	12·50	500	25	750	37·50	1000	50

The Value of Gold per Troy Ounce at Different Degrees of Fineness, based on $20·6718 per Ounce for 1,000 Fine.

Fine.	$	Cents.	Fine.	$	Cents.	Fine.	$	Cents.	Fine.	$	Cents.
10	0	20·67	260	5	37·47	510	10	54·26	760	15	71·06
20	0	41·34	270	5	58·14	520	10	74·94	770	15	91·73
30	0	62·02	280	5	78·81	530	10	95.61	780	16	12·40
40	0	82·69	290	5	99·48	540	11	16·28	790	16	33·07
50	1	03·36	300	6	20·16	550	11	36·95	800	16	53·75
60	1	24·03	310	6	40·83	560	11	57·62	810	16	74·42
70	1	44·70	320	6	61·50	570	11	78·29	820	16	95·09
80	1	65·37	330	6	82·17	580	11	98·97	830	17	15·76
90	1	86·05	340	7	02·84	590	12	19·64	840	17	36·43
100	2	06·72	350	7	23·51	600	12	40·31	850	17	57·11
110	2	27·39	360	7	44·19	610	12	60·98	860	17	77·78
120	2	48·06	370	7	64·86	620	12	81·65	870	17	98·45
130	2	68·73	380	7	85·53	630	13	02·33	880	18	19·12
140	2	89·41	390	8	06·20	640	13	23·00	890	18	89·79
150	3	10·08	400	8	26·87	650	13	43·67	900	18	60·46
160	3	30·75	410	8	47·55	660	13	64·34	910	18	81·14
170	3	51·42	420	8	68·22	670	13	85·01	920	19	01·81
180	3	72·09	430	8	88·89	680	14	05·68	930	19	22·48
190	3	92·76	440	9	09·56	690	14	26·36	940	19	43·15
200	4	13·44	450	9	30·23	700	14	47·03	950	19	63·82
210	4	34·11	460	9	50·90	710	14	67·70	960	19	84·50
220	4	54·78	470	9	71·58	720	14	88·37	970	20	05·17
230	4	75·45	480	9	92·25	730	15	09·04	980	20	25·84
240	4	96·12	490	10	12·92	740	15	29·72	990	20	46·51
250	5	16·80	500	10	33·59	750	15	50·39	1000	20	67·18

Expansion of Metals Through Heat.

Bodies.	Expansion in 0° to 100° C.	
	Decimals.	Fractions.
Lead	0·002848	1–351
Iron, wrought	0·001167	1–856
Iron, cast	0·001110	1–901
Glass, white	0·000861	1–1,161
Gold	0·001552	1–645
Copper	0·001717	1–582
Marble	0·000849	1–1,178
Brass	0·001920	1–521
Platinum	0·000856	1–1,167
Sandstone	0·001174	1–852
Silver	0·001909	1–524
Steel, hardened	0·001225	1–816
,, soft	0·001079	1–926
Zinc, rolled	0·003331	1–302
Zinc, cast	0·002987	1–336

Table of Heating Degrees.

		C.	F.
Red	commences at	525°	955°
Dark Red	,, ,,	700°	1,292°
Cherry Red	,, ,,	850°	1,562°
Light Red	,, ,,	900°	1,652°
Yellow	,, ,,	1,100°	2,012°
White	,, ,,	1,300°	2,372°
Full White	,, ,,	1,500°	2,732°

Prospector's Gold Table for determining the Value of Free Gold per Ton (2,000 lb.) of Quartz or Cubic Yard of Gravel.

The table below furnishes an exceedingly simple method for determining the value of free gold in a ton of gold-bearing quartz or a cubic yard of auriferous gravel.

Take a sample of four (4) pounds of quartz, pulverize it to the usual fineness for horning; wash it carefully by batea, pan, or by other means; amalgamate the gold by the application of quicksilver; volatilize the quicksilver by blow-pipe or otherwise; weigh the resulting button, and the value given in the table opposite such weight will be the value in free gold per ton of 2,000 lb. of quartz.

Example:—Sample of four lb. produces button weighing one grain, the fineness of the gold being 830; then the value of one ton of such quartz will be $17·87.

If the sample of four pounds should produce a button weighing, say, four-tenths of a grain (0·4) then the value of such quartz would be (830 fine) $7·14 per ton.

Weight washed gold 4 lb. sample. Grains.	Fineness 780. Value per oz. $16·12	Fineness 830. Value per oz. $17·15	Fineness 875. Value per oz. $18·08	Fineness 920. Value per oz. $19·01
0·1	$1·68	$1·78	$1·88	$1·98
0·2	3·36	3·57	3·76	3·96
0·3	5·03	5·36	5·65	5·94
0·4	6·71	7·14	7·53	7·92
0·5	8·40	8·93	9·42	9·90
0·6	10·07	10·73	11·30	11·88
0·7	11·75	12·51	13·19	13·86
0·8	13·43	14·29	15·07	15·84
0·9	15·11	16·08	16·95	17·82
1	16·79	17·87	18·84	19·81
2	33·59	35·74	37·68	39·62
3	50·38	53·61	56·52	59·43
4	67·18	71·49	75·36	79·24
5	83·97	89·36	94·20	99·05

To determine the gold value of a cubic yard of auriferous gravel, the same table can also be used.

Take a sample of sixty (60) pounds of gravel, pulverize it, and carefully wash it by batea, pan or otherwise; amalgamate the gold, volatilize the quicksilver; weigh the button, and in column in table, opposite the weight, will be found the gold value of the cubic yard of gravel.

Example :—Sample of sixty pounds produces button weighing one grain, the fineness of the gold being 780; then the value of one cubic yard of such gravel would be $1·67. This is arrived at by pointing off one point, or dividing the value given in the table by 10. If the sample of sixty pounds yields a button weighing five-tenths (0·5) of a grain, then the value of the gravel would be—gold being 780 fine—$0·84 per cubic yard.

Simple Ore Tests.

The following simple tests will show whether an ore carries any precious metals. Afterwards samples of the rock should be assayed to ascertain the amount of value per ton.

Gold.—Powder; roast if sulphurets are present; grind very fine and wash in pan or spoon; examine with lens; yellow particles not soluble in nitric acid. The colour of pure gold is bright yellow, tinged with red. Gold may be distinguished from all other metals or alloys by the following simple traits: It is yellow, malleable, not acted upon by nitric acid.

Silver.—Pure silver is the brightest of metals, of a beautiful white colour and rich lustre.

Chloride of Silver.—If suspected in a pulp, harshly rub a bright and wet copper cartridge thereon. If a chloride or chloride-bromide of silver, it will whiten the copper. Graphite will thus whiten copper or gold, but can be rubbed off.

Copper.—After roasting the pulp, intimately mix and well knead with a like quantity of salt and candle grease or any other fat, and cast into the fire, when the characteristic colours—first blue, then green—will appear. This test is better made at night.

Galena.—Black zinc blende is often mistaken for galena. The two may be distinguished by the infallible sign: the powder of galena is black; that of blende, brown or yellow.

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